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METHODS AND COSTS OF STRIPPING AND MINING AT THE UNITED VERDE OPEN-PIT MINE, JEROME, ARIZONA.

BY

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METHODS AND COSTS OF STRIPPING AND MINING AT THE UNITED VERDE OPEN-PIT MINE, JEROME, ARIZ.[1]

By E. M. J. Alenius[2]

INTRODUCTION

This paper describing the mining practices at the United Verde open-pit mine at Jerome, Ariz., is one of a series being prepared by the United States Bureau of Mines on mining practices, methods, and costs in the various mining districts in the United States.

The purpose of this paper is to present the methods used in developing and operating the mine, and to discuss the problems encountered.

The United Verde open-pit mine is situated in Bitter Creek Gulch on the east slope of the Black Hills, directly west of Jerome, in the central part of Arizona. The elevation of the surface is 5,490 feet above mean sea level. West of the mine the slope of the hills continues to the flat-topped Mingus Mountain which reaches an altitude of 7,720 feet; to the east the hills slope towards the Verde River which has an elevation of 3,350 feet at Clarkdale.

ACKNOWLEDGMENT

Mr. J. C. Perkins, superintendent of the steam-shovel department of the United Verde Copper Co., rendered valuable assistance in the preparation of this paper by supplying information relative to the open-pit operations.

HISTORY

Although the United Verde mine has been operated since 1876, open-pit mining with power shovels was not begun until the fall of 1918. Shovel operations had been preceded by glory-hole mining on the south end of the ore body. The upper portion of the ore body was originally mined by underground methods. However, in 1893 a fire occurred in the upper workings, which necessitated the abandonment of several high-grade stopes and increased the cost of mining in the adjacent areas. Later subsidence took place, seriously affecting the smelter and

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2 One of the consulting engineers, U. S. Bureau of Mines.

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surface plant which had been constructed on top of the ore body (fig. 1). For 25 years efforts were made to mine the ore in the fire areas and to reduce the fire, but without success. At the end of this time Mr. R. E. Tally, general manager of the United Verde Copper Co., devised an open-cut method for mining the ore in the fire zone.

In 1920 stripping was advanced to permit the removal of a small amount of schist ore. During 1921 and for three months in 1922 operations were suspended on account of the low price of copper. Since 1922 mining has rapidly advanced, and the major stripping operations were completed in October, 1927.

Yardage removed up to January 1, 1929

Year	Stripping, cubic yards	Ore, cubic yards	Total, cubic yards
1919	374,260		374,260
1920	1,026,382	5,760	1,032,142
1922	486,260	132,481	618,741
1923	882,524	361,912	1,244,436
1924	1,359,645	142,740	1,502,385
1925	1,653,009	165,084	1,818,093
1926	1,323,991	210,479	1,534,470
1927	570,869	230,007	800,876
1928	322,538	276,055	598,593
Total, Jan. 1,1929	7,999,478	1,524,518	9,523,996

GEOLOGY

The general geological features at Jerome are well exposed on the rough slopes of the Black Hills. The summit of the Black Hills is covered by Tertiary basalt resting upon horizontal Paleozoic limestone and some sandstone. Next below lie the pre-Cambrian rocks, which are very evident at Jerome. These rocks consist chiefly of the so-called greenstone complex. Intruded in the greenstone are masses of rhyolite porphyry and diorite. The United Verde ore body lies in massive pyrite which is between adjacent masses of the rhyolite porphyry and diorite. The pyrite is a replacement deposit of schist, formed in the overlying concave trough of the United Verde diorite.

The Verde fault cuts through Jerome below the exposed pre-Cambrian rocks in a general northwesterly direction. The fault has a throw of about 1,700 feet. North and east of Jerome, on the other side of the fault and at a lower elevation than the pre-Cambrian and Paleozoic beds, the Tertiary basalt is again apparent, covering the successive rocks. Towards the Verde River these downthrown rocks are more or less covered with Quaternary gravels.

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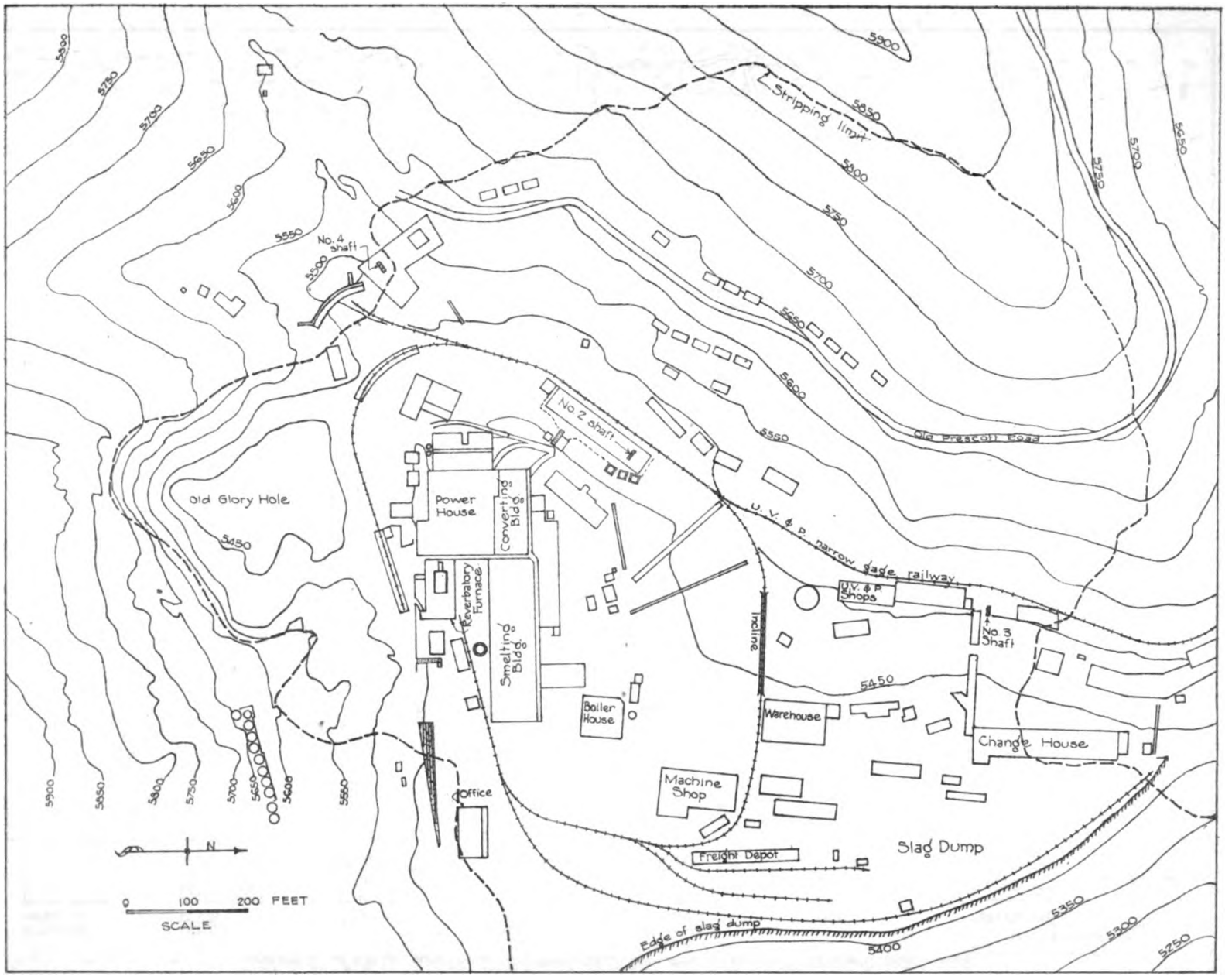

FIGURE 1.- PLAN OF UNITED VERDE SURFACE PLANT AND SMELTER. BEFORE STEAM SHOVEL OPERATIONS

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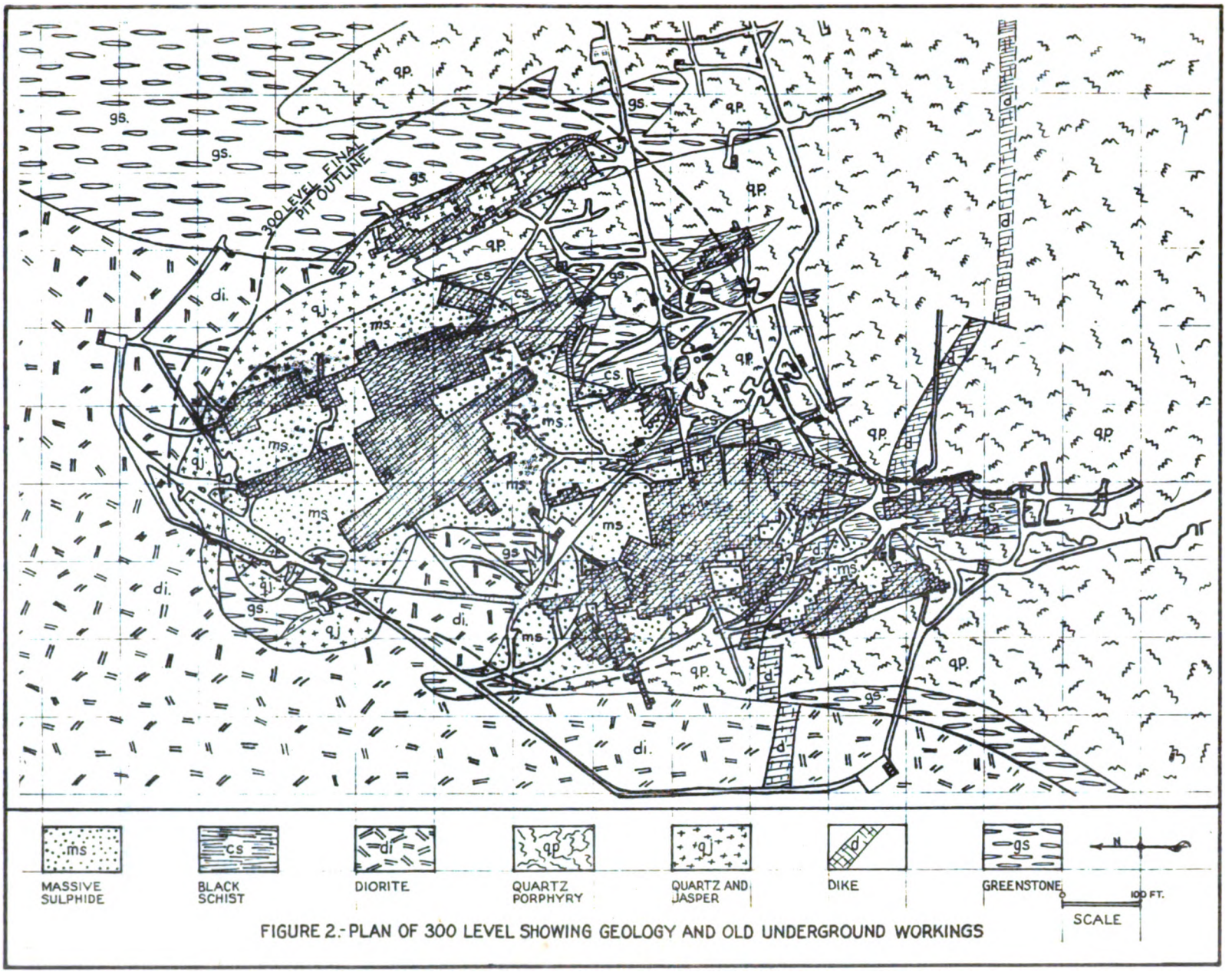

FIGURE 2.- PLAN OF 300 LEVEL SHOWING GEOLOGY AND OLD UNDERGROUND WORKINGS

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ORE DEPOSIT

The massive-sulphide ore body, consisting mostly of pyrite with associated copper and zinc sulphide, is a pipe-like mass resting on the quartz-rhyolite porphyry footwall and supporting a dense barren diorite hanging wall. The zone containing copper ore is approximately 1,000 feet by 800 feet in plan on the 300-foot level (fig. 2). It dips steeply to the northwest and has been opened to the depth of 3,000 feet. The upper portion of the sulphide ore body has undergone considerable oxidation, which extends roughly to the 160-foot level. In the oxidized zone the ore consists chiefly of copper oxides and quartz carrying gold and silver. Below the zone of oxidation the massive sulphides are enriched with the secondary copper minerals, chiefly chalcocite. On the footwall side of the ore body the massive sulphides are more or less separated from the porphyry by sulphide-bearing schist shoots extending into the quartz porphyry. The massive sulphide is low in silica adjacent to the schist, and increases in silica as the diorite is approached. It is generally separated from the diorite by barren quartz and jasper.

The ore material is in general divided into three classes--direct smelting, concentrating, and converter. The direct-smelting ore consists chiefly of the massive sulphides. Clean mineralized schist and siliceous sulphide are sent to the concentrator, whereas mineralized material containing 60 per cent of silica or better is used for converter flux. Some material obtained in stripping operations which is too low grade for milling or smelting is piled for leaching.

PRELIMINARY PREPARATION AND PLANS

Before shovel operations could be started it was necessary to select new sites for the smelter and mine surface plant. A new smelter was constructed west of the Verde River at Clarkdale, 5 miles below Jerome on the Atchison, Topeka & Santa Fe Railway, and began operating in 1915. Shortly afterwards the present mine plant was built at the elevation of the 500-foot level, about 1/2-mile north of the main workings, and an adit was run on that level into the main ore body for the transportation of men and material. New shafts with underground hoist rooms and headframes also had to be completed before the entire upper portion of the ore body became completely available for the pit operations. For the removal of ore from the underground mine the Hopewell Tunnel was run on the 1,000-foot mine level to cut the slope of the hills about 1¼ miles north of the mine. The Verde Tunnel & Smelter Railroad was constructed to transport ore from the portal of the tunnel to the smelter at Clarkdale and later was extended to Jerome so that the heavy equipment necessary for the pit operations could be brought up to Jerome.

The first plan of operations called for exposing the entire ore body on the 300-foot level and extending the walls above that level into the porphyry and diorite waste on a one-half to 1° slope. Below the 300-foot level a part of the ore body was to be mined to the 400-foot level by glory-holing. Accordingly, operations were started at the intersection of the 300 level with the slope, at the point closest to the ore body. At the same time switchbacks on the side of

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the diorite hill were constructed so that the upper part of the bank might be removed on a series of benches approximately 50 feet apart. Work was conducted on two of the upper benches around the level of the old smelter and mine plant. The upper bench would be at an elevation of 5,250 feet or about 640 feet above the 300-foot level. Mining was carried on according to this plan until the shutdown in 1921.

From geological data obtained in the underground workings it was estimated that approximately 1,900,000 cubic yards or 4,000,000 tons of ore remained above the 400-foot level. In order to make this available the original plan required the removal of 14,000,000 cubic yards of stripping. During the suspension of operations the original plans were revised. Instead of cutting an approach on the 300 level, the cut was to be made on the 160 level, and glory-hole operations were to be carried from the 160 level to the 500 level. This change of plan was deemed advisable on account of the fact that the 160 level is roughly the dividing level between the zones of oxidation and secondary enrichment, and also because the change would decrease the total yardage to be stripped. The yardage would be further decreased by steepening the slope of the pit wall. At this time it was believed that the material of the final bank would stand on a one-half to 1° slope. Berms were planned for the final wall, two to be 20 feet wide and one 30 feet wide on the 160 level. These berms were to serve for drainage and for keeping the banks trimmed. At the resumption of operations in 1922 it was estimated that 7,000,000 cubic yards of stripping remained to be removed to make available 2,400,000 cubic yards of ore.

Thus two distinct methods of mining were employed at the United Verde open pit. The first operation extended above the 160 level and consisted chiefly of stripping and removing the oxidized ore by steam shovels working on benches. The second extended between the 160 level and 500 level and was confined chiefly within the ore zone. The original revised plan was to mine the ore by means of mill holes, but conditions arising later made necessary a revision of the plans. However, the two methods of operation are entirely distinct, and in continuing this paper, methods and costs for the upper-pit or major stripping operations will be discussed in detail separately from the lower-pit operations. Yardages involved in the two divisions are as follows:

Upper and lower pit yardage

	Removed to Jan. 1, 1929	Balance remaining	Total yardage
Major stripping operations:			
Ore	803,354	- -	803,354
Waste	7,274,097	- -	7,274,097
Total	8,077,451	- -	8,077,451
Lower-pit operations:			
Ore	721,164	2,121,822	2,842,986
Waste	725,381	796,357	1,521,738
Total	1,446,545	2,918,179	4,364,724
Grand total	9,523,996	2,918,179	12,442,175

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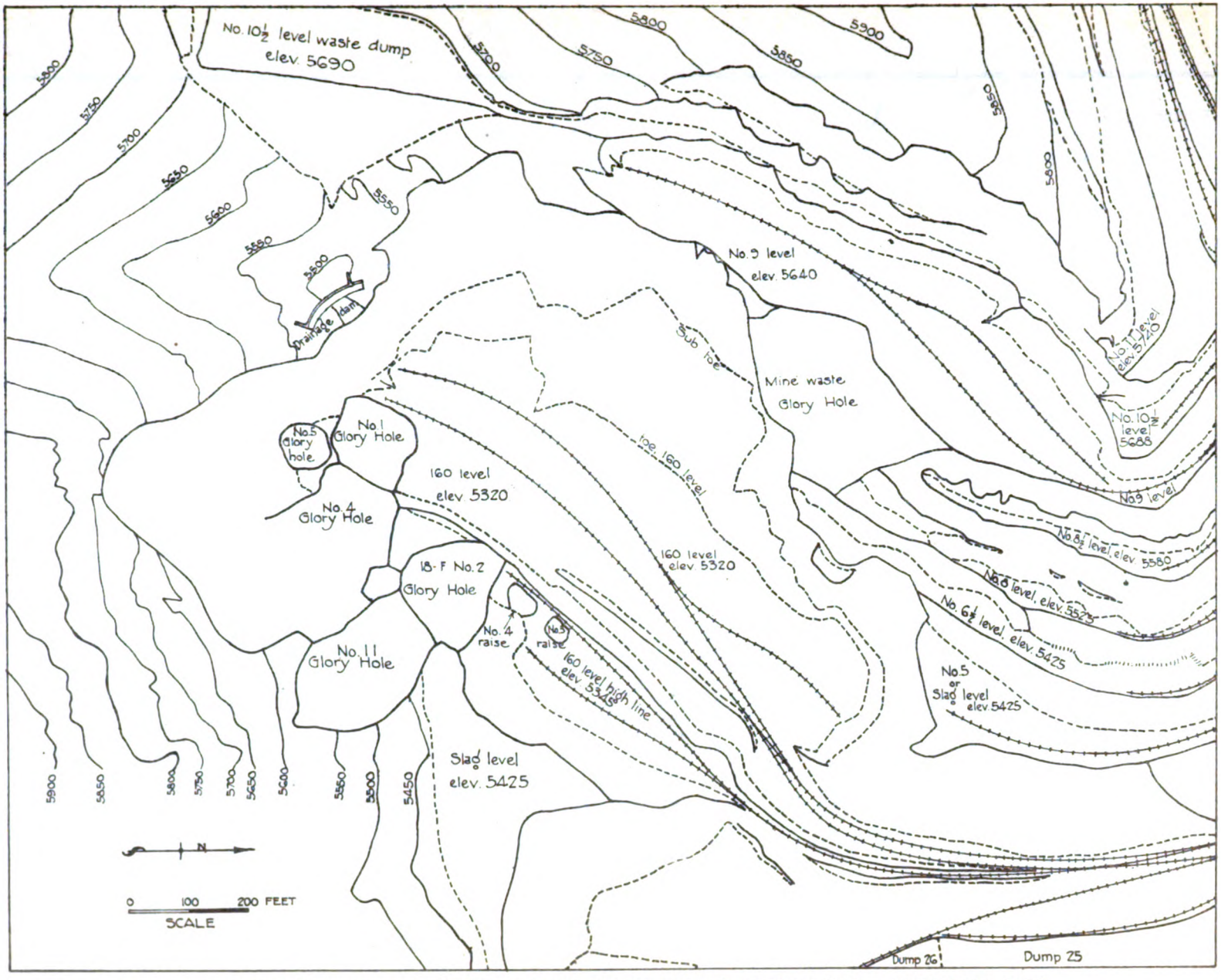

FIGURE 3.–SURFACE PLAN OF UNITED VERDE MAJOR STRIPPING OPERATIONS, JANUARY 1, 1925

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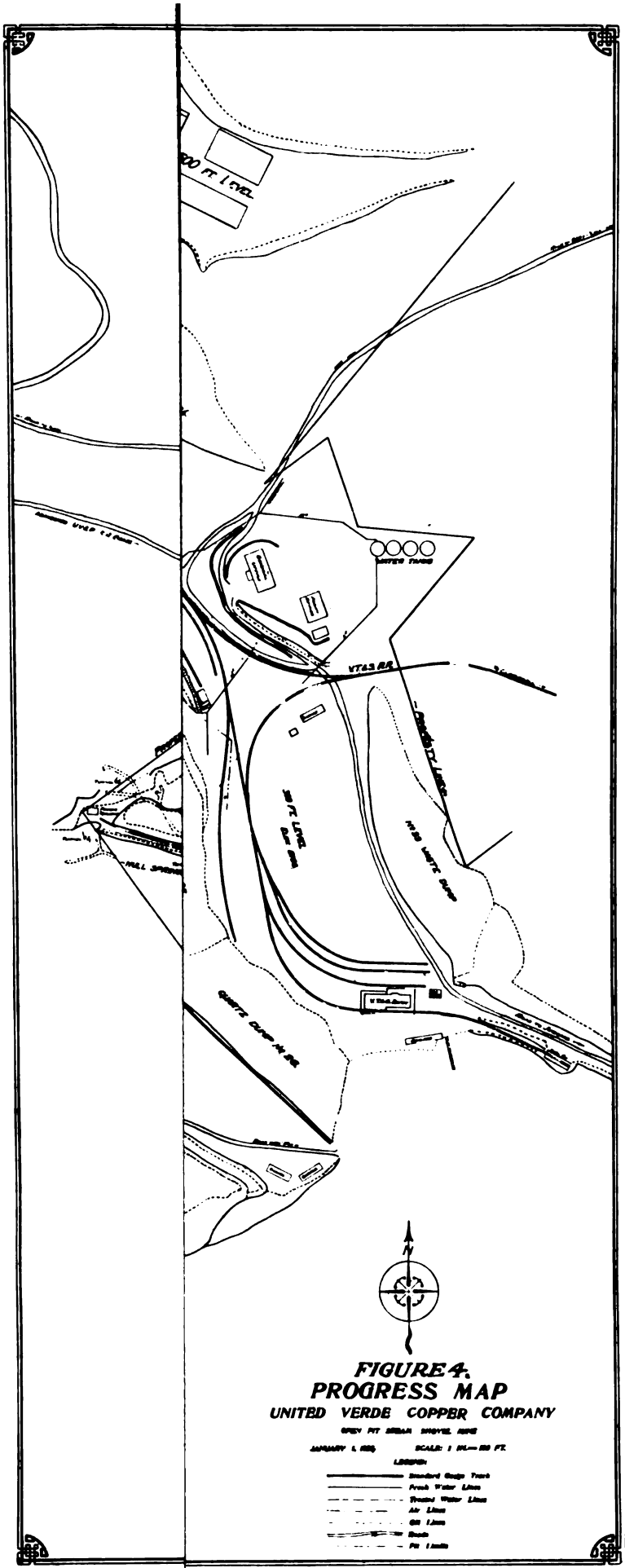

FIGURE 4.
PROGRESS MAP
UNITED VERDE COPPER COMPANY

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MAJOR STRIPPING OPERATIONS

General

The major stripping operations were completed in October, 1927. The following review of the work done and the problems encountered is presented.

Ore above the 160 level consisted chiefly of oxides and quartz. Sulphide and schist ore were mined to a slight extent. The dense augite diorite was the main material stripped. On the surface the diorite was weathered and was easily broken down. Greenstone, porphyry, low-grade oxides and quartz, and approximately 1,600,000 cubic yards of slag made up the balance of the stripping material.

Development of Upper Pit

The upper pit had eight major levels (fig. 3). The bottom or 160 level had a bank 110 feet high, the next six above had banks approximately 50 feet high, and the bank on the top level varied in height from a few feet to 140 feet. Standard-gage railroad tracks connected each level by means of a system of switchbacks on the adjacent side of the hill (fig. 4). The switchbacks were planned with tailtracks on the pit end connecting with the levels. Waste dumps were maintained along the switchbacks and in nearby gulches.

Switchback Construction and Cost

In order to work on the various benches as planned, 12 standard-gage switchbacks were required between the 300 level, elevation 5,200, and upper bench, elevation 5,740 feet. Ninety-pound rails were laid on the four switchbacks between the 300 level and the 160 level, and 75-pound rails were laid on the upper switchbacks. Seventy-five pound rails were also employed on the benches for the loading track. Sixty-pound rails were used on the dumps. The maximum grade was 3 per cent. Material was hauled downgrade to the dumps. However, when it later became necessary to move material from the 160 level to a dump at a higher elevation, the grade on the fifth level was reduced to 2 per cent. The maximum curvature was 24°. All curves were tapered and compensated for grade. No. 7 switches were used exclusively. At one time approximately 7 miles of standard-gage track was being maintained.

General data on the switchbacks is tabulated as follows:

Total length of switchbacks	27,064 feet
Cost per mile excavating and grading	$42,700
Cost per cubic yard excavating and grading	$1.41
Cost per mile track material (75-lb. rail)	$18,000
Cost per mile laying track	$4,000

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Equipment

Rolling stock used during the major stripping operation is tabulated as follows:

Rolling stock used in major stripping

No. used	Item	Type	Weight, size, or capacity
1	Steam shovel	Full revolving	8-cu. yd. dipper
2	Steam shovel	Standard	4-cu. yd. dipper
1	Steam shovel	Full revolving	¾-cu. yd. dipper
5	Steam locomotives	Switcher	82.5 tons
2	same	Switcher	53.5 tons
30	Cars	Air dump	25-cu. yds.
1	Spreader	- -	14-ft. spread
1	Locomotive crane	- -	100 tons
2	Track shifters	Peterson	
6	Gondolas	F. B. composite	50-ton
6	Flat cars	- -	50-ton
1	Tank car	- -	50-ton, 10,000-gals.

The full-revolving shovel was mounted on traction wheels running on parallel tracks laid with 130-pound rails. The 4-yard standard railroad shovels were first used on sections of standard-gage track and were supported rigidly by jack arms. These shovels were later equipped with caterpillar traction trucks, which provided greater flexibility and eliminated jacking delays.

The full-revolving shovel was placed on the lower bench of the 160 level. It was planned to make the lower bench 110 feet high, as that was the difference in elevation between the 160 level and the top of the slag dump. The 4-yard shovels were used on the upper benches. It was originally intended that work on the upper benches should be completed in sufficient time to permit the large shovel to remain on the 160 level and maintain a 110-foot bench. The small 3/4-yard shovel was used for excavation on switchbacks and roads.

The crew on each steam shovel except the small one consisted of an engineer, a craneman, a fireman, an oiler, and two to four pitmen.

The switcher-type locomotives were used for tramming. They were equipped with water leg-type tenders of 4,500 gallons capacity. The 82.5-ton locomotives were able to haul six loaded cars and the 53.5-ton locomotives could haul four cars. The locomotive crew consisted of an engineer, a fireman, and a switchman.

Both steam shovels and locomotives were originally equipped for coal burning, but later they were altered to burn oil, as it was found that a saving of 28.9 per cent of the cost per cubic yard trammed was made by using oil for fuel. The average cost of coal per ton for the period compared was $10.08 against $2.045 per barrel of fuel oil; 2.98 barrels of fuel oil was equivalent to 1 ton of coal. Superheated steam was also found to be advantageous for shovels and locomotives.

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The 25-cubic yard air-dump cars were used for handling the loaded rock. Occasionally large pieces were loaded which would not pass through the raised door when dumping and which had to be blasted out.

The spreader was used for spreading the dumped material so that the track could be shifted.

The track shifters were particularly valuable for shifting track on the dumps as well as for loosening the track from the ballast when reclaiming.

The crane was equipped with clamshell and orange-peel buckets for handling coal and other material, as well as for excavating and grading. The crane was also used to assist in the heavy repair work on the shovels and cars and to pick up cars that were wrecked and turned over on the dumps.

Gondola cars were required for hauling coal and loose material. When the oil-burning equipment was installed the fuel oil was transported to the shovels in a tank car. Flat cars were used to handle construction material, repair parts for the shovels, drill steel, and blasting supplies.

Buildings and Shops

The buildings and shops consisted of the warehouse, office, locomotive house, machine shop, blacksmith shop, and car-repair shop. Storage tanks and a sand dryer were also required.

Water System and Treatment Plant

Sufficient water was obtained from several springs on the east slope of the Black Hills, and was distributed by gravity through pipes on the various benches. Treated water for distribution on the two upper levels was pumped to a storage tank above the top bench. A lime-soda-ash softener was first installed to treat the water, but did not eliminate foaming in the boilers. The foaming was due to a large amount of calcium sulphate present in the raw water which was changed to sodium sulphate in the treatment process. To eliminate this foaming an automatic lime-barium carbonate water softener with a capacity of 4,000 gallons hourly was installed. The hardness of the raw water, which varied from 14 to 30°, was reduced to 3°, chiefly by the elimination of the mineral constituents. The cost of treating the water by the lime-barium process averaged $0.066 per 1,000 gallons.

Drilling and Blasting

Drilling and blasting has always been the most serious problem at the United Verde pit. During the early stages of mining the ground was easily broken on the upper benches. The altered diorite was quickly prepared for blasting. Churn-drilling was mainly used during this period, with holes spaced 35 to 40 feet apart at a distance of 5 to 10 feet back of the edge. These holes were generally drilled 12½ per cent deeper than the height of the bank, and were loaded with 10 to 12 boxes of 35 per cent strength gelatin. Toe holes were used occasionally for removing hard ribs.

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The favorable conditions did not persist. Below the altered zone the diorite was fresh, very hard, and dense, besides being blocky and fissured. The rate of churn-drilling dropped from 15 to 20 feet per shift to 3 to 5 feet per shift, and on account of the slips and seams occurring in the diorite, difficulty was experienced in keeping the holes straight. Churn-drilling was then abandoned and replaced by drilling with air machines. A combination of toe-holing with jack-hammers and drilling vertical holes with tripod-mounted drifters was finally adopted (fig. 5) after toe-holing alone proved impractical. Shooting with toe holes alone caused the bank to be undercut and did not secure fragmentation, which resulted in delaying the shovels while bowlders were being blasted. A general comparison of costs during these three periods is as follows:

Comparison of costs of drilling and blasting methods

	Churn-drilling, per cu. yd.	Toe-holing, per cu. yd.	Combination method, per cu. yd.
Labor	$ .0467	$ .0631	$ .1132
Explosives	.0259	.0438	.0683
Total	.0726	.1069	.1815
Average cubic yards per 8-hour shift loaded by shovels..	800	530	758

On the lower bench or 160 level the slag and material from the oxidized zone were removed very rapidly, whereas progress on the upper levels was slow because of the difficulty of breaking the unaltered diorite; this disarranged the schedule of operations. As the large shovel crowded the work on the upper benches, a consolidation of the benches on the west end resulted, so that on the west side of the pit the height of the bank above the 160 level reached 315 feet. In order not to shut down the large shovel on the 160 level or move it to an upper level the ground beneath this high bank had to be broken. Coyote blasting was then used to break the ground. By this time most of the material from the oxidized zone had been removed, and the material in the bank was all barren diorite. Large blasts were therefore practical, as not much of the material would become mixed. Six coyote blasts were set off to break material on the 160 level.

Two powder foremen, each assisted by a jigger boss, supervised the drilling and blasting; one foreman worked on the lower bench and the other on the upper benches. The drilling and blasting crew on the lower bench consisted of about 25 men. On the upper benches 50 to 60 men were needed.

The principal methods of breaking ground at the United Verde upper pit--churn-drilling, combination toe-hole and vertical drilling, and coyote blasting--will be considered in detail.

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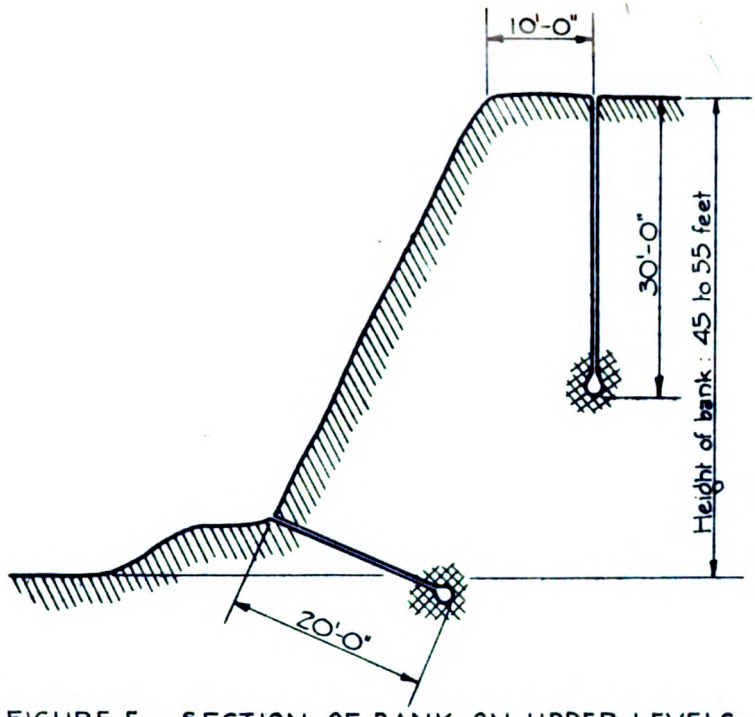

FIGURE 5.- SECTION OF BANK ON UPPER LEVELS SHOWING VERTICAL AND TOE HOLES

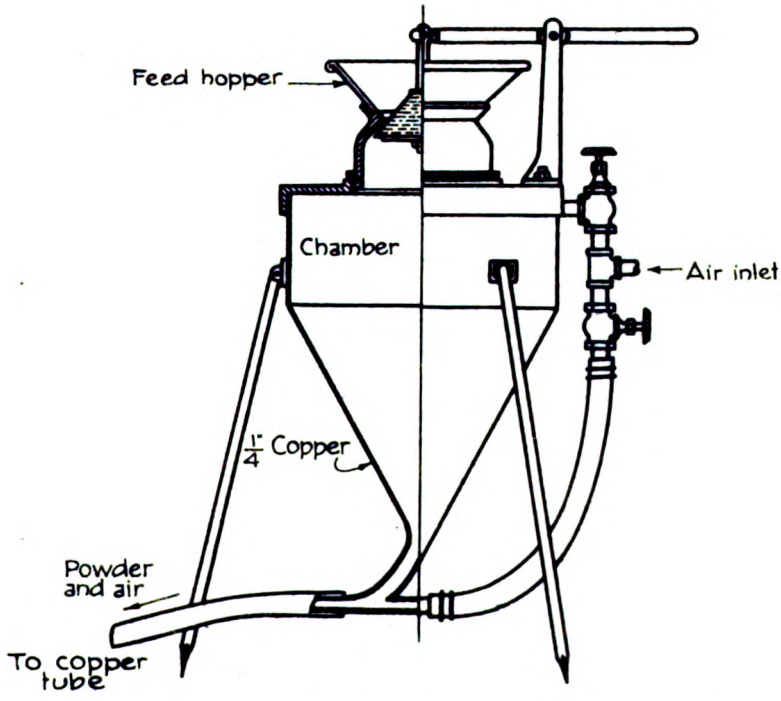

FIGURE 6.- UNITED VERDE POWDER LOADING MACHINE

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Churn Drilling and Blasting on the 160 Level. - For breaking the softer ground on the lower level three steam churn drills were mainly used. Six-inch holes were drilled 10 to 15 feet deeper than the height of the bank. Holes were cased with 6-inch pipe in loose ground. The drilling speed varied from 5 to 30 feet per shift, depending on the character of the material. The holes were placed close to the edge of the bank and spaced from 30 to 50 feet apart. The distances from the bottom of the hole to the toe of the cut varied from 55 to 95 feet. These distances and the character of the ground governed the size of the powder charge, which ordinarily ranged from 4,000 to 8,000 pounds. In a few instances the burden on the churn-drill holes was reduced with toe holes, but on account of inability to trim the bank thoroughly, this was considered unsafe.

Hot ground was encountered in certain areas on the 160 level which were churn-drilled. The temperature in some of these holes varied from 120 to 240° F. It was necessary to cool these holes before loading. In some cases if the hole remained for two or three days before loading it would cool off sufficiently to permit loading; in others it was necessary to run a stream of water into the hole for as much as 24 hours or more. In all holes 35 per cent strength gelatin dynamite was used, as it was considered less sensitive to deterioration from the heat of the rock. A temperature of 120°F. was considered safe for loading. To permit the men to keep away from the collar of the hole when loading, a wooden trough was designed by which powder and sand tamping could be pushed into the hole from a distance. Generally four No. 8 electric caps were placed in the large holes and fired from a 440-volt line. Seldom more than three holes were shot at a time. The cost of churn-drilling for 1926 is as follows:

	Per foot
Labor	$1.97
Supplies	.65
Fuel	.68
Repairs	.98
Miscellaneous ...	.26
Total	$4.54

Average footage drilled was 9.2 feet per shift.

A drill crew consisted of three men - a driller, a fireman, and a helper.

Combination Machine Drilling and Blasting. - The general data referring to breaking ground by machine drilling and blasting is tabulated as follows:

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Breaking ground by machine drilling and blasting

Item	Toe holes	Vertical holes
Kind of drill used	Jackhammers	Large Leyners
Depth of hole	18-20 ft.	21-30 ft.
Feet drilled per hour	3.5	3.7
Drilling crew	1 man	2 men
Labor cost per foot	$0.22	$0.42
Percentage of holes lost ..	20	35
Spacing of holes in line ..	6-15 ft.	12.5 ft.
Powder charge	100-250 lbs.	75-150 lbs.
Kind of powder	50 per cent gelatin and Quarry Special No. 1	Quarry Special No. 1 and 50 per cent gelatin
Steel used	7/8-in. quarter-octagon	1¼-in. hollow-round
Gage of steel, starter	2-1/4 in.	2-3/4 in.
Gage of steel, finisher ...	19/16 in.	21/16 in.
Length of starter	2-1/2 ft.	2-1/2 ft.
Length of finisher	20 ft.	30 ft.
Number of changes	12	19

Jackhammers, which are light and easy to handle, were used for drilling toe holes. In drilling vertical holes the Leyners were suspended by a block and tackle from a tripod made from 2-inch pipe. On the heavy machines two men were required to handle the larger steel. When two parallel rows of vertical holes were shot at the same time the holes were staggered.

An attempt was made to drill toe holes with Leyners mounted on tripods and also with heavier jackhammers. However, on account of the time required to set up, the greater loss when one of the larger holes had to be abandoned in the fissured ground, and the extra labor required, the use of the heavier machines was abandoned in favor of the light jackhammers.

Compressed air was obtained from the main mine compressors on the 500 level and was delivered to the pit at 85 pounds pressure through a 4-inch pipe line about one-half mile long.

Both vertical and toe holes were heavily chambered to receive the required amount of explosives. The quantity of explosives used in chambering the holes varied from 10 to 15 per cent of the charge. During the first stage of operations the chambered holes were loaded with stick explosive, and the cartridges were pushed into the chamber with a wooden rod. This method was slow, particularly in fractured ground; also the miners were inclined to use unsafe methods in forcing the cartridges through the ragged portions of the holes that resulted from springing.

Experiments were made in loading granulated powder with compressed air, and the powder loader shown in Figure 6 was designed and adopted. Whereas it required 20 minutes to load 50 pounds of stick explosive, the same amount of granulated powder could be loaded in 60 seconds. The powder loader consisted of

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a sealed copper chamber fitted with a hopper through which the powder was placed in the chamber. Compressed air was then admitted into the chamber; the powder was forced down into a flexible rubber tube, from which it was carried by a jet of compressed air through a copper loading tube. This loading tube was placed in the hole to be loaded, the end of the tube resting a few inches from the bottom of the chamber. As the chamber filled up, the tube was withdrawn. The experienced operator was enabled to determine when the chamber had been filled by the change in rate of withdrawing the tube. The chamber of the loader was designed to hold 50 pounds of granulated powder. Quarry Special No. 1 powder was found best suited for use in this machine. Before being placed in the container, the powder was put through a 4-mesh copper screen, so that no lumps large enough to clog the loader would be admitted.

Air loading of granulated powder is generally considered unsafe by the powder manufacturers. However, during the three years the method was used at the United Verde pit not even a slight mishap occurred, whereas several serious accidents resulted from the former method of loading.

No. 8 electric detonators were used exclusively for detonating the charges. Two caps were placed in each hole, one previous to the charge and the other after the chamber was about three-fourths filled. Current at 440 volts from the main power line was used for igniting the detonators. Between 20 and 30 holes were fired at one time.

Although the combination-method toe and vertical holes gave better fragmentation, an enormous quantity of bowlders was still present in the blasted rock. If it was possible for the shovel to set the bowlders aside, they were later block-holed. In many instances the bowlders had to be "dobied" before the shovel could operate. Approximately 20 per cent of the total powder used was for secondary blasting.

Coyote Blasting. - Coyote blasting was confined to the 160 level. Six of these large blasts were fired during the major stripping operations. The high banks that were not accessible for churn drilling could be broken by means of toe holes with a fair degree of success in soft ground. In hard diorite, however, the banks could not be broken in this manner. A round of toe holes drilled with considerable danger beneath the high bank on the west side of the pit merely undercut the bank and furnished less than half a shift's digging for the large shovel. The fresh diorite in high banks was also too hard to be broken by means of churn-drill holes. In order to break the high bank with safety and in sufficient quantities, coyote blasting was deemed to be the only practical method.

The development work of the coyote blasts consisted of driving the necessary adits and sinking the powder pockets at the desired distances from the bank. For the first three blasts, work was started from the face of the bank. However, this required that the bank be cleaned up before development for another blast could be started. For the last three blasts an old adit behind the bank was utilized, and as soon as one blast had been fired development for a new blast started. The drifts were $4\frac{1}{2}$ by 6 feet in cross section. Powder pockets were

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5 by 5 feet in section, and were sunk to a depth of 20 to 30 feet below the pit grade at intervals of 30 feet in the drifts. The distance from the pockets to the face of the cut varied from 78 to 135 feet in the different blasts. Upon the completion of the development work, the pockets were wired up and the lead wires protected by being encased in old rubber hose. Four No. 8 electric detonators in a case of gelatin dynamite were placed in the bottom of each pocket. The powder was unloaded from box-cars at the mouth of the tunnel and transported to the pockets on specially constructed wooden cars. After the charge was loaded into the pockets, the powder was covered with paper and sand tamping. Back-filling of the tunnel was started as soon as the pockets had been loaded.

After the first coyote blast, heat radiating from the mine fire and resulting high humidity were encountered in one of the end pockets. The rock temperature in this pocket was 152°F. In the other pockets the heat varied from 112 to 124°F. During development work it was necessary to cool the drift by means of blowers and piping. Tests on explosives and detonators were made under the conditions existing in the end pockets. It was found that black powder absorbed moisture and became useless; 35 per cent strength gelatin dynamite lost some sensitiveness, and 50 per cent gelatin dynamite appeared unaffected. The tar covering on the electric caps was melted, but the cap tested satisfactorily, as did Cordeau Bickford. To overcome the moisture, the pockets and drift adjacent to the hottest ground were gunnited. This did not reduce the temperature. Before loading the powder charges the pockets were lined heavily with asbestos to keep the powder from direct contact with the hot ground.

Black powder was used in the first three blasts. The first blast was in the softer material along the contact, and was considerably overshot. The second and third blasts were undershot, as the black powder did not have sufficient shattering force to break the hard ground effectively. To obtain better shattering of the rock, Quarry Special powders with a higher disruptive effect were used. The speed was further increased by the addition of higher explosives; 60 per cent strength ammonia dynamite was used in the last blast. The ground was satisfactorily broken in the last three blasts with a considerable decrease of secondary blasting. Figure 7 shows the plan of blast 6 and Figure 8 shows the section through No. 9 pocket for blast 6.

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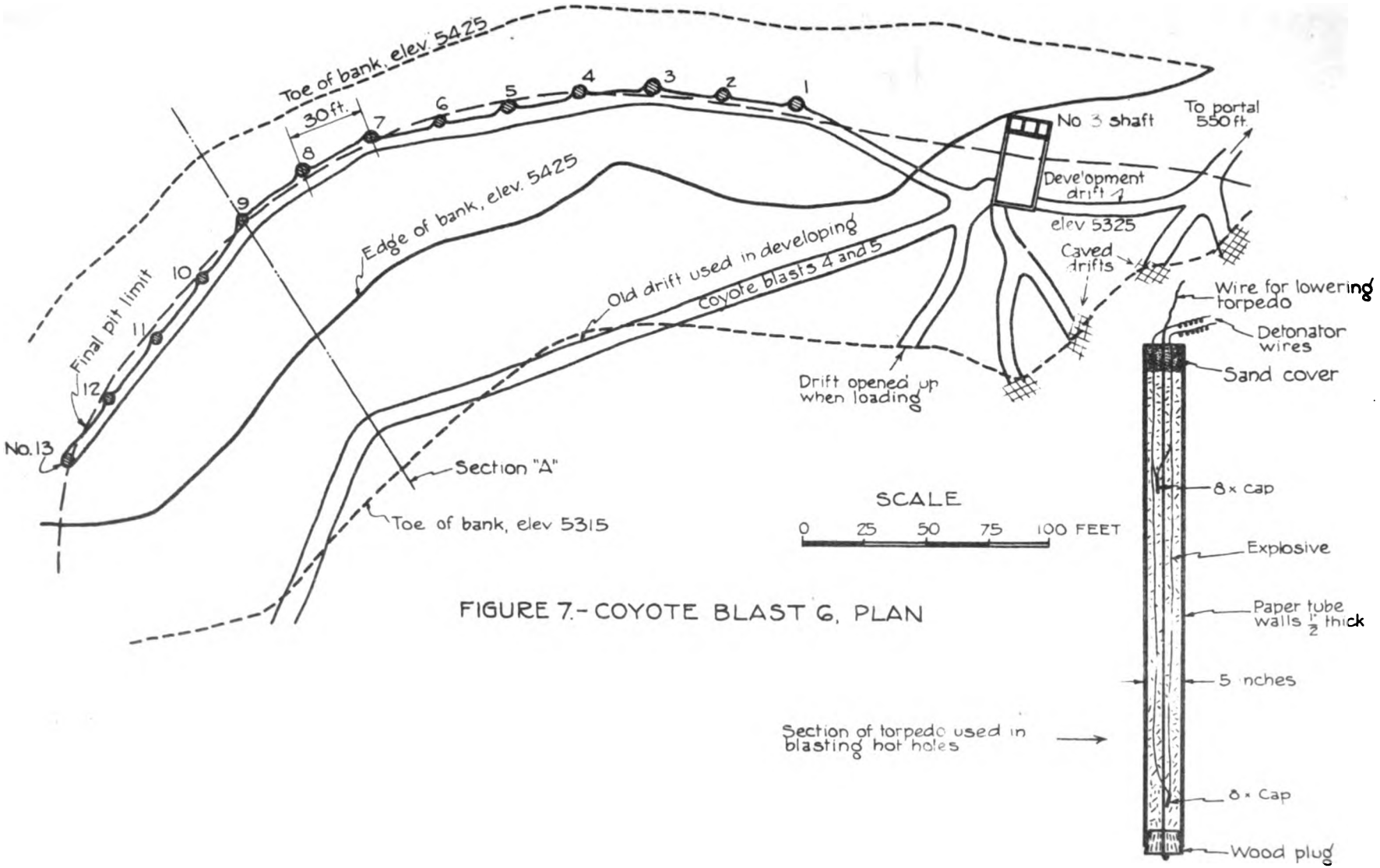

FIGURE 7.- COYOTE BLAST 6, PLAN

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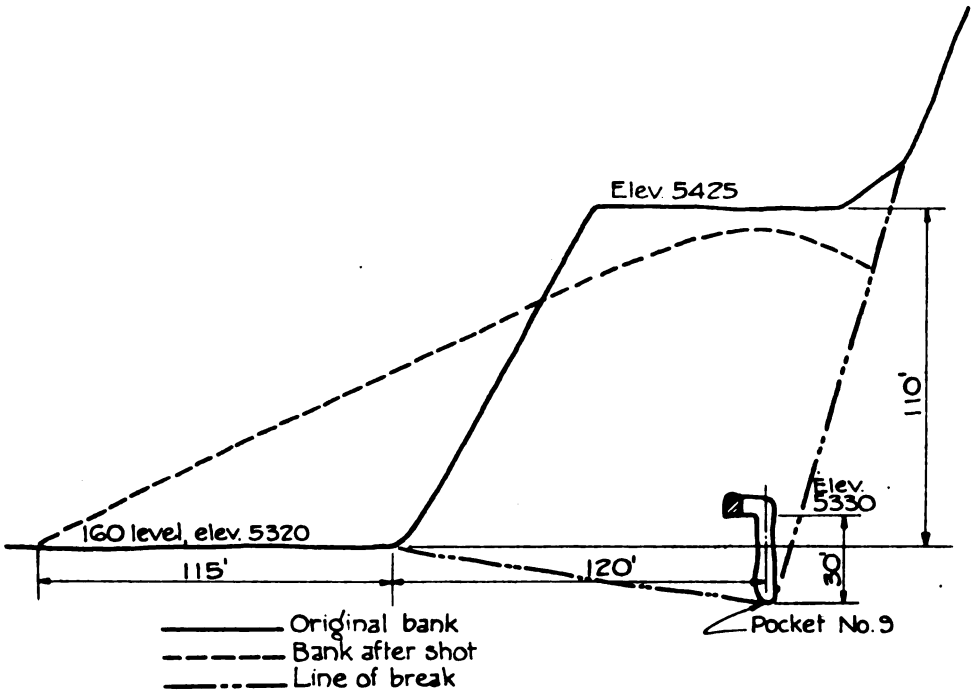

FIGURE 8.–SECTION "A" THROUGH POCKET NO. 9, COYOTE BLAST 6

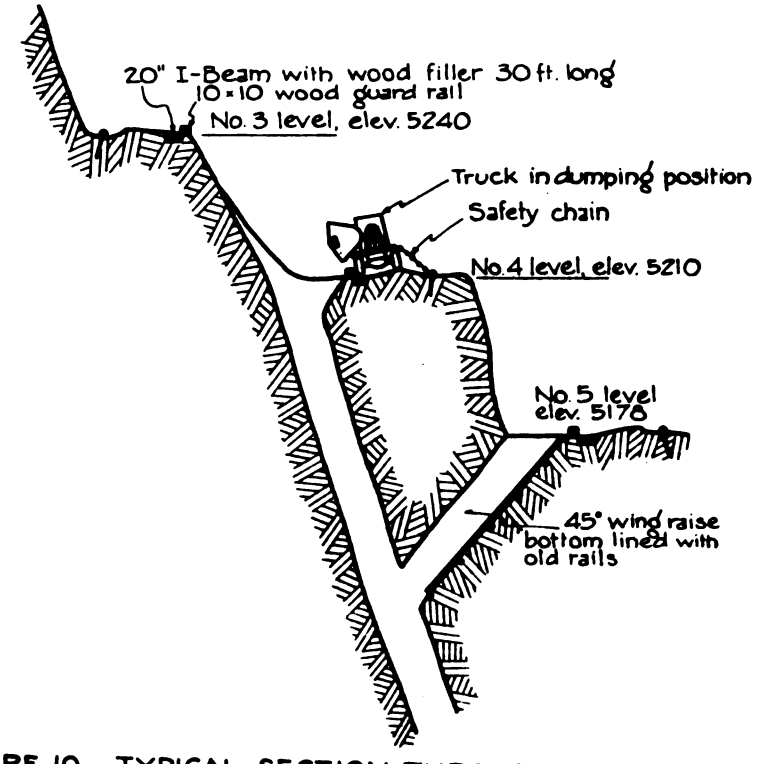

FIGURE 10.– TYPICAL SECTION THROUGH NO. 3 RAISE SHOWING DUMPS ON THREE LEVELS

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Table 1 gives detailed information on each of the six coyote blasts.

Table 1. - Coyote blasting data

	Blast No. 1	Blast No. 2	Blast No. 3	Blast No. 4	Blast No. 5	Blast No. 6
Number of pockets	7	6	14	10	5	13
Average distance between pockets, feet	30	30	30	30	30	30
Total development drifting and sinking, feet	410	485	910	686	225	800
Average distance of pocket to toe of bank, feet	78	100	133	65	61	100
Average vertical height above pocket, feet	121	167	189	146	120	120
Average height of bank, feet	(1)	(1)	(1)	(1)	100	100
Average burden on each pocket, cu. yds.	5,000	7,800	9,800	6,000	5,230	9,240
Black powder used, pounds	100,000	95,400	250,000	- -	- -	- -
Quarry Special No. 6 used, pounds	- -	- -	- -	91,700	- -	200,000
Quarry Special No. 4 used, pounds	- -	- -	- -	- -	45,000	- -
60 per cent ammonia dynamite used, lbs.	- -	- -	- -	- -	5,000	30,000
50 per cent gelatin dynamite used, lbs.	13,050	3,500	9,900	11,650	- -	3,250
35 per cent gelatin dynamite used, lbs.	- -	- -	- -	- -	4,000	- -
Total pounds of powder used	113,050	98,900	259,900	103,150	54,000	233,250
Cubic yards broken	140,000	135,000	259,000	105,000	54,000	200,000
Cubic yards per pound of explosives	1.24	1.36	1.00	1.02	1.00	0.86
Cubic yards per foot developed	394	278	285	153	240	250
Costs per cubic yard:						
Development, labor and supplies	$0.0850	$0.0827	$0.0502	$0.1297	$0.1400	$0.0407
Explosives	0.0783	0.0697	0.0915	0.1065	0.1360	0.1272
Total cost per cubic yard	0.1633	0.1524	0.1418	0.2362	0.2760	0.1679

1 Pockets under slope only.

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Shovel Operation

On the 160 level the first cut with the large shovel was parallel to the face of the slag dump, and the cars were loaded on the opposite side from the bank and on a higher grade than the cut. After a considerable amount of slag had been removed, the shovel was headed directly into the ore body and a thorough cut made through the entire extent. The loading track was carried on one side of the cut about 15 feet above the level of the shovel, and ground was filled in on one side so that the track could be extended as the cut advanced. After the first cut was completed the shovel began widening out on the north and west sides. As the cut became wider it was necessary to shift the loading track to the same level as the shovel.

On the upper benches where the slope of the ground was not too steep a narrow cut was made on the grade of the bench and track was laid for loading. Ground to be loaded by the 4-yard shovels could be broken down without covering the track; the level was established as soon as the first cut was completed.

When the slope was too steep, it was necessary for the 4-yard shovels to cast the broken material over the bank to make room for the track and to establish the level. This material was picked up on a lower level, generally the 160. As soon as a level was established, the shovel was moved back to one end. Ground was broken ahead of the shovel, which advanced the entire length of the bench. After each cut, the loading track was shifted to parallel the face of the cut. With the 4-yard shovels the top level was completed first and lower benches were completed successively.

Two bench foremen had direct charge of the work. One foreman looked after the shovel and locomotive operations in the pit and on the dumps of the 160 bench. The other foreman had charge of the two shovels on the upper benches and of the tramming operations. On account of the short haul to the switchback dumps on the upper benches, a switch tender was not required. However, on the 160 bench a switch tender was required at the entrance to the pit and also on the dump approach, in order to control the movement of trains.

During the time all three shovels were in operation two shifts were maintained - a day shift from 7.30 a.m. to 5 p.m. and a night shift from 6 p.m. to 2 a.m. After the 4-yard shovels were removed from the upper benches, the 300 shovel was worked on three 8-hour shifts.

Disposition of Material. - On the upper benches the material consisted chiefly of diorite, with a small amount of low-grade quartz ore on the two benches above the 160 level. Most of the ore was obtained on the 160 level. The material encountered was classified and disposed of as follows:

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Classification and disposition of material

Material	Grade	Class	Destination
Quartz	Over $1.25 in Ag and Au	Converter ore	To smelter or stock pile for smelter.
Quartz	Below $1.25 in Ag and Au	Possible converter ore	To stock pile.
Oxide	Over $6	Direct smelting ore	To smelter or stock pile.
Oxide	Over $1.25, below $6		To stock pile.
Oxide	Below $1.25	Leach ore	To leach dump.
Sulphide	Over 3% copper	Direct smelting ore	To smelter.
Sulphide	Over 1%, below 3% Cu	Possible smelting ore	To stock pile.
Sulphide	Below 1% copper	Leach ore	To leach dump.
Schist	Over 3% copper	Direct smelting ore	To smelter.
Schist	Over 1%, below 3% Cu	Possible milling ore	To stock pile.
Schist	Below 1% copper	Leach ore	To leach dump.
Diorite, slag, greenstone, porphyry	Barren	Waste	To waste dump or underground mine for filling.

The first material shipped to the smelter was transported directly from the pit to the smelter over the Verde Tunnel & Smelter Railroad. The Hopewell haulage tunnel constructed on the 1,000-foot level ran through the footwall of the ore body. By dropping the ore from the pit through raises and hauling it out of the tunnel it was estimated that a saving of 31 cents a ton could be made in transporting material from the pit to Hopewell. Therefore two raises were completed in 1922, one for direct-smelting ore, the other for converter ore. These raises were brought up in the footwall adjacent to the pit. A track was extended along the sides of the raises so that the loaded cars could be dumped directly into them. On the 900 level the raises were equipped with grizzlies and blasting chambers for reducing the larger material. Between the 900 and 1,000 foot levels, storage bins of 1,000-ton capacity were cut out and provided with pneumatic doors and chutes. Forty-ton standard-gage cars are used in the Hopewell tunnel.

During the major stripping operations the United Verde mill had not been completed. As most of the schist encountered was of high grade it was shipped as direct-smelting ore.

Stock piles were established at convenient locations adjacent to the pit and at the mouth of Hopewell tunnel. Material which could not be smelted at the time and low-grade material which might have future value were placed on these stock piles. At Jerome dump 25 was made for the higher-grade oxides and dump 24

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for the lower-grade oxides. High-grade quartz was placed on Hopewell dump 34. Lower-grade quartz was placed on Jerome dump 26 and Hopewell dump 33. The low-grade sulphide stock pile was at Hopewell.

A waste glory hole was maintained on the surface in the diorite and connected with a raise for the distribution of waste to be used as filling in the underground mine. This waste-transfer raise intersected with the pit levels. The underground mine was assured of a sufficient quantity of waste for filling the stopes as long as the open-pit operations continued. However, the waste delivered to the underground mine represented a very small part of the total waste handled, being about 150,000 cubic yards per year. Part of the waste was placed on the switchback dumps, and the switchback tracks were moved out from the hill as the dumps were built up. Most of the waste was placed on four dumps filling up adjacent gullies. The height of the dumps varied from 50 to 300 feet. The average haul was about 4,000 feet and the maximum 7,000 feet.

The proper destination of the material was determined by bank samples obtained before the ground was blasted and by samples taken of the broken material. If classification of the material was doubtful, the cars that had been loaded were set out until the assay was returned. All cars were grab-sampled after loading so that a record could be obtained of the material, especially of that which was placed on the stock piles. All pit samples were sent to the main assay office for assaying.

Costs of Major Stripping Operation

The cost of stripping was deferred to be absorbed later as the ore was made available. That is, the ore later recovered would carry a charge for removing the waste above it. To the direct charge for mining, an estimated charge of $1.50 was added to pay for the stripping removed.

Direct operating costs for 1925 are given in Table 2.

It may be noted in the summary of direct operating costs that the charge for repairs is very high. The United Verde pit probably has harder rock to handle than any other pit. Work in this hard rock resulted in severe strains to the digging equipment, which accounts for the high repair cost.

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Table 2. - Summary of costs

United Verde major stripping operations — Year 1925

	Cu. yds.
Total stripping removed in 1925	1,647,469
Total ore removed in 1925	165,084
Total material removed in 1925	1,812,093
Dry tons of ore mined in 1925	272,005 tons

Direct operating costs per dry ton ore mined and per cubic yard of stripping

	Labor	Comp. air drill and steel[1]	Power cost	Fuel cost	Explosives	Repairs[2]	Supplies	Misc.	Total
Stripping (per cu. yd.)									
Drilling and blasting .	$.1012	$.0361		$.0029	$.0871		$.0100	$.0037	$.2410
Loading	.0541		$.0005	.0430		$.1083	.0061	.0022	.2142
Transportation	.0550			.0447		.1011	.0024	.0081	.2113
Dump maintenance ..	.0381	.0002		.0006		.0016	.0057	.0122	.0584
Total stripping.	.2484	.0363	.0005	.0912	.0871	.2110	.0242	.0262	.7249
Mining (per ton)									
Drilling and blasting .	.0693	.0234		.0008	.0459		.0073	.0024	.1491
Loading	.0403		.0030	.0165		.0528	.0036	.0013	.1175
Transportation	.0349			.0177		.0437	.0058	.0015	.1036
Total mining ...	$.1445	$.0234	$.0030	$.0350	$.0459	$.0965	$.0167	$.0052	$.3702

1 Repairs to drills included.
2 Includes labor and repair supplies.

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General operating data for 1925 are tabulated as follows:

Shovel operation

	4-cubic yard shovels	8-cubic yard shovels
Total 8-hour shifts	1,244.6	747.5
Cubic yards per shift	784	1,001
Cubic yards per barrel of fuel oil	64.7	50.8
Barrels of fuel oil per shift	12.1	19.7
Per cent time of loading	46.4	49.1
Per cent time of operating delays	38.6	36.9
Per cent time of general delays	15.0	14.0

Locomotive operation

Total 8-hour shifts	4,485.3
Cubic yards per shift	357.6
Cubic yards per barrel of fuel oil	46.7
Barrels of fuel oil per shift	7.66

Blasting

Pounds explosives per cubic yard broken	0.648

Miscellaneous

Total man-shifts	154,191
Cubic yards (ore and stripping) per man-shift	11.79
Total labor cost per cubic yard	$.4081
Repair cost (labor, supplies, power, etc.) per cu. yd.	$.2325

LOWER PIT OPERATIONS

Operations below the 160 level are confined chiefly within the main ore body. The entire area of the ore body is exposed on the 300-foot mine level. Below the 300 level, ore will be left in the pit wall to be removed later, and between the 300 level and the 160 level considerable stripping will have to be removed in order to maintain a safe working slope. In the harder material an attempt is being made to maintain a 60° slope, while in the softer material a 45° slope is necessary. Scarcely any oxide remained below the 160 level. The massive sulphides form the chief ore material, and on the sides of the sulphides, mineral-bearing schist for milling and quartz for converter ore are obtained. Low-grade greenstone, porphyry, schist, jasper, and barren diorite are more or less on the margin of the ore body. Throughout the mass of the ore body underground workings are encountered. The old square-set stopes are generally filled. However, in some areas the old stopes were abandoned when the fire broke out, and

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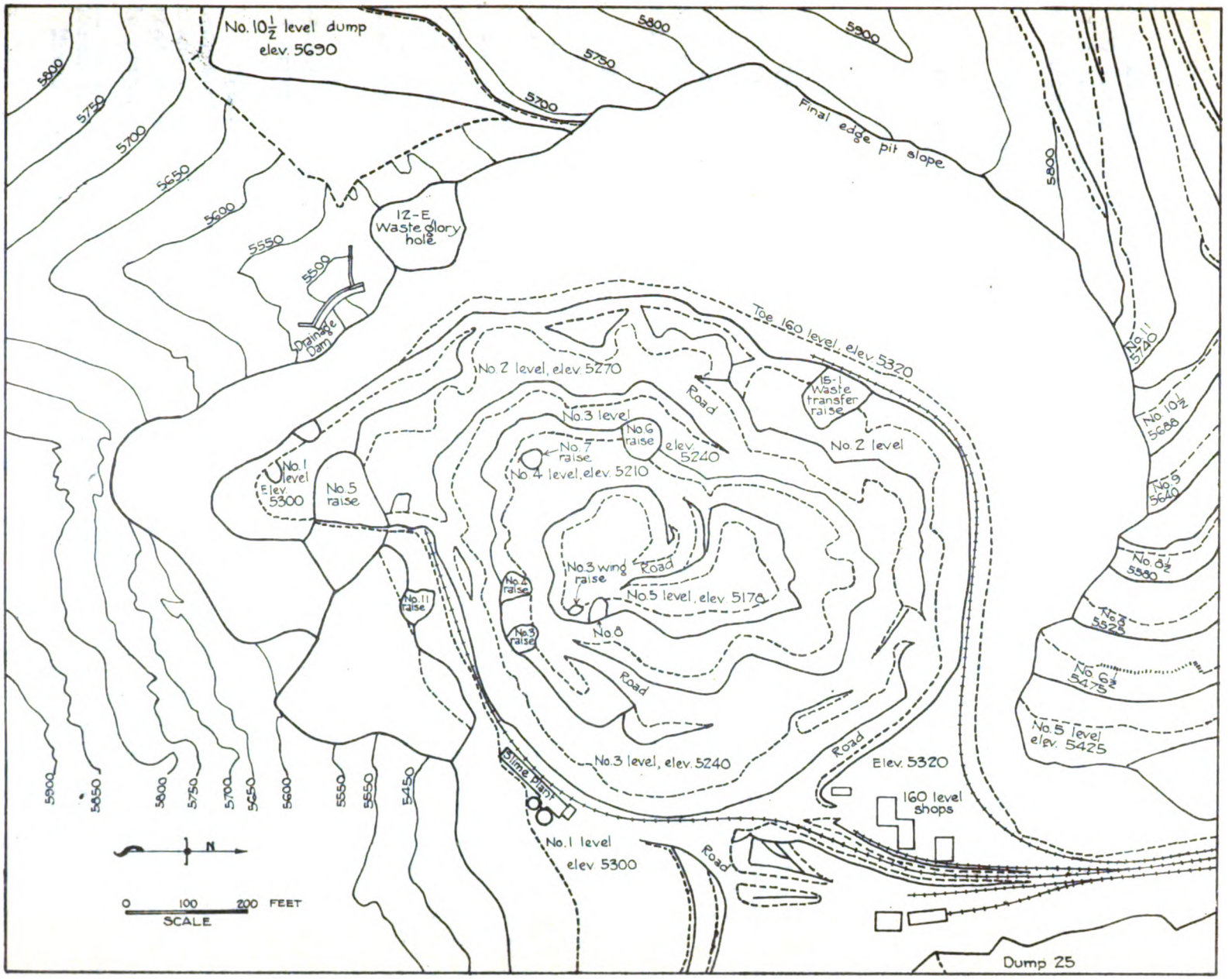

FIGURE 9.- PLAN OF UNITED VERDE PIT, JULY 1, 1929

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the upper floors could not be filled. In operating in the lower pit, these old workings have to be considered in order to avoid mishaps.

Development

As has been noted before, the original plan for removing material below the 160 level was by glory-holing. In 1924, after a portion of the ore body on the 160 level bench had been exposed and made accessible, several glory holes were started. However, they did not prove practical for the following reasons:

(a) Inability to drive raises in the desired locations on account of the underground fire and existing stopes.

(b) Dilution of ore and inability to sort material properly.

(c) Indefinite production.

(d) Operating hazards.

The United Verde mine fire extended more or less through the entire ore body above the 800 level, except in the extreme south end. Although certain parts of the ore body were accessible on various levels, the driving of a continuous raise from the 1,000-foot level was practically impossible except in the footwall. In the small glory holes that were started in the sulphide area, transfer of material on intermediate levels was necessary to avoid the fire areas.

In the several attempts to drive raises throughout the main ore body it was necessary to open the fire areas which had been sealed off. In these areas water had to be sprayed on the timbers in a few instances to prevent combustion. Double crews had to be maintained to keep the work going, and it was necessary to have cool air blowing on the men continuously. In several places the heat of the ground necessitated wrapping the gelatin dynamite in asbestos when loading holes. Moreover in certain areas the burnt ground would not stand up, as the sulphides had been reduced to a powder by the action of the fire. After a few attempts were made and a serious accident occurred by the igniting of the dust when blasting,[3] further work was abandoned.

In driving raises through the old stopes it was impossible to hold the dried-out filling satisfactorily. Raises therefore had to be driven through the solid ground.

The ore body of the United Verde mine is very complex. High-grade sulphides, schist, and quartz are adjacent to each other or to low-grade materials. High and low grade areas also occur in the mass of sulphides. Glory-holing was started around No. 3 and No. 4 raises (fig. 9). Alternate seams of high and low grade porphyry and schist surrounded the collars of these raises. In blasting the

3 Gardner, E. D., and Stein, Edmund, Explosibility of Sulphide Dusts in Metal Mines: Repts. Investigations, Serial 2863, Bureau of Mines, 1928, 11 pp.

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material the different grades became mixed, as it was impossible to blast advantageously in the confined limits of the glory holes. If it was possible to blast different materials separately, the destination of the material as it was removed on the 1,000-foot level was changed to correspond with the different classes. However, part of one class of material stuck to the sides of the raise and in the bin, causing a dilution of the material as the raise was changed to high or low grade material. In addition, the separation of stope fill presented difficulties, and the old timbers encountered had to be pulled out of the glory hole in order not to clog the raises and chutes.

No definite production could be depended upon in a system of glory holes, as the material available would be more or less controlled by the working condition of the various glory holes. It would be impossible to work a certain glory hole faster in order to obtain the desired material in that area without interfering with work in adjacent glory holes. Greater difficulties in keeping the connecting raises clean and open would also be expected, as large rocks would be thrown into the raises when blasting.

Glory-holing is more or less hazardous because men have to work on slopes around loose rock. In the United Verde pit men would also be subjected to the added hazard of working under high walls on the sides of the pit and of not being able to leave the working place conveniently if loose material sloughed off of the high bank. Also above the fire area men would have to work on hot ground through which gas and smoke seeped from the mine fire below.

These conditions prevented the mining of the lower pit by glory-holing, but in the early part of 1925 a plan was devised by Mr. W. V. DeCamp, now assistant general manager of the United Verde Copper Co., by which the lower pit would be mined by small shovels working on low benches. The materials were loaded into trucks which dumped into any one of several transfer raises extending to the 1,000-foot mine level. The disadvantages of the glory-hole method were entirely overcome by this change.

Several additional raises were constructed in the porphyry footwall away from the fire area. An old mine shaft was cleaned out and utilized. Except for cutting around a transfer raise it is possible to use one raise constantly for one class of material, and dilution in the raises is practically eliminated. By the use of trucks operating on the levels, material can be transported conveniently from any part of a level to any raise. Figure 10 shows the arrangements for dumping into a transfer raise from three different levels. A plan to use cars operating on narrow-gage track was given up as it did not provide maximum flexibility. Steep grades connecting the various levels are necessary, and it is possible for motor trucks to haul from one level to another.

The small shovels are able to sort material very well. The shovels can also be moved around so that drilling and blasting can be done advantageously. The big problem in the lower pit operations is the sorting of the ore. This is first accomplished by proper drilling and blasting, and later by sorting and thorough cleaning up of loose material when digging. Softer material is generally

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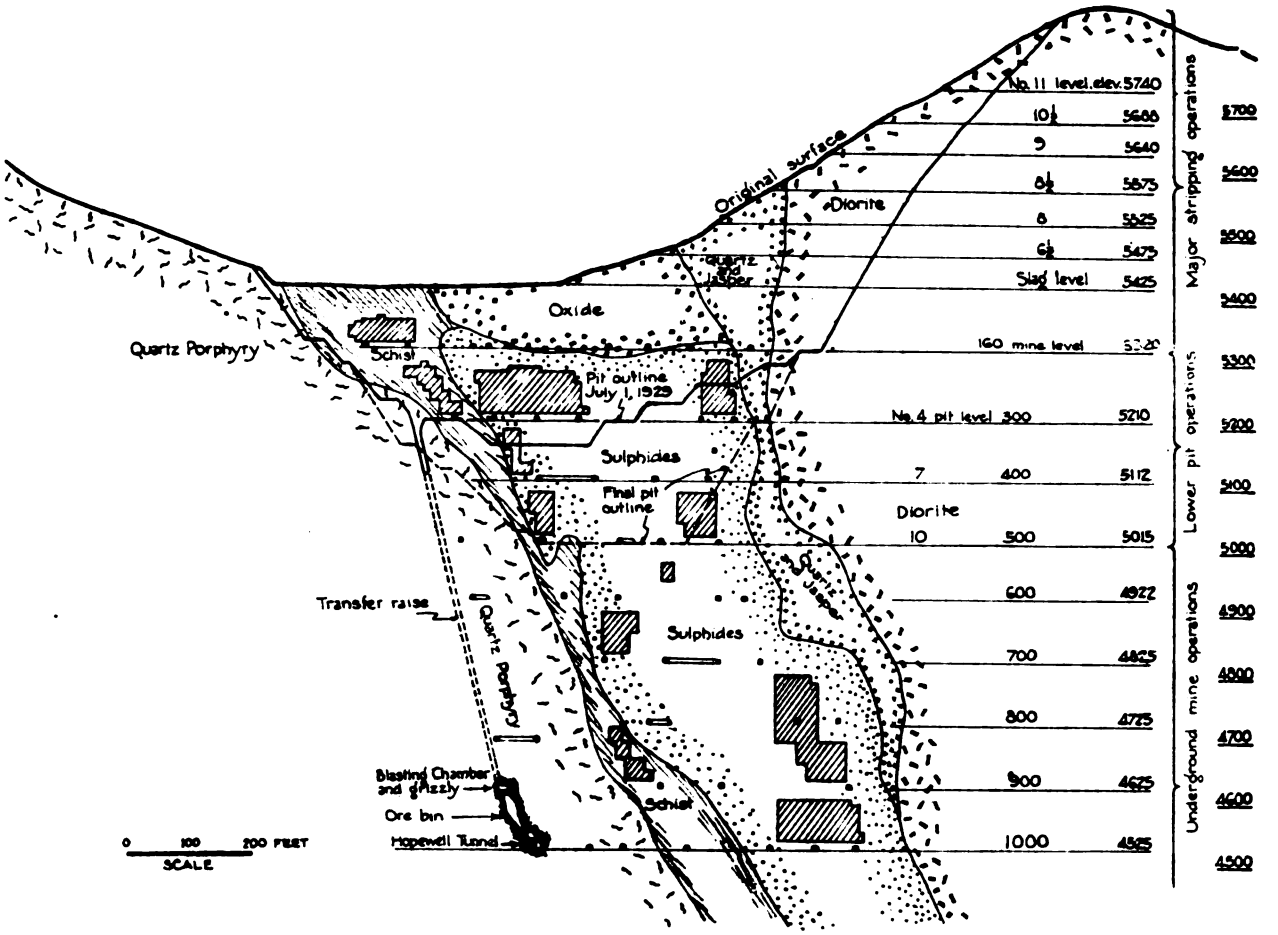

FIGURE 11.- TYPICAL SECTION OF UNITED VERDE PIT

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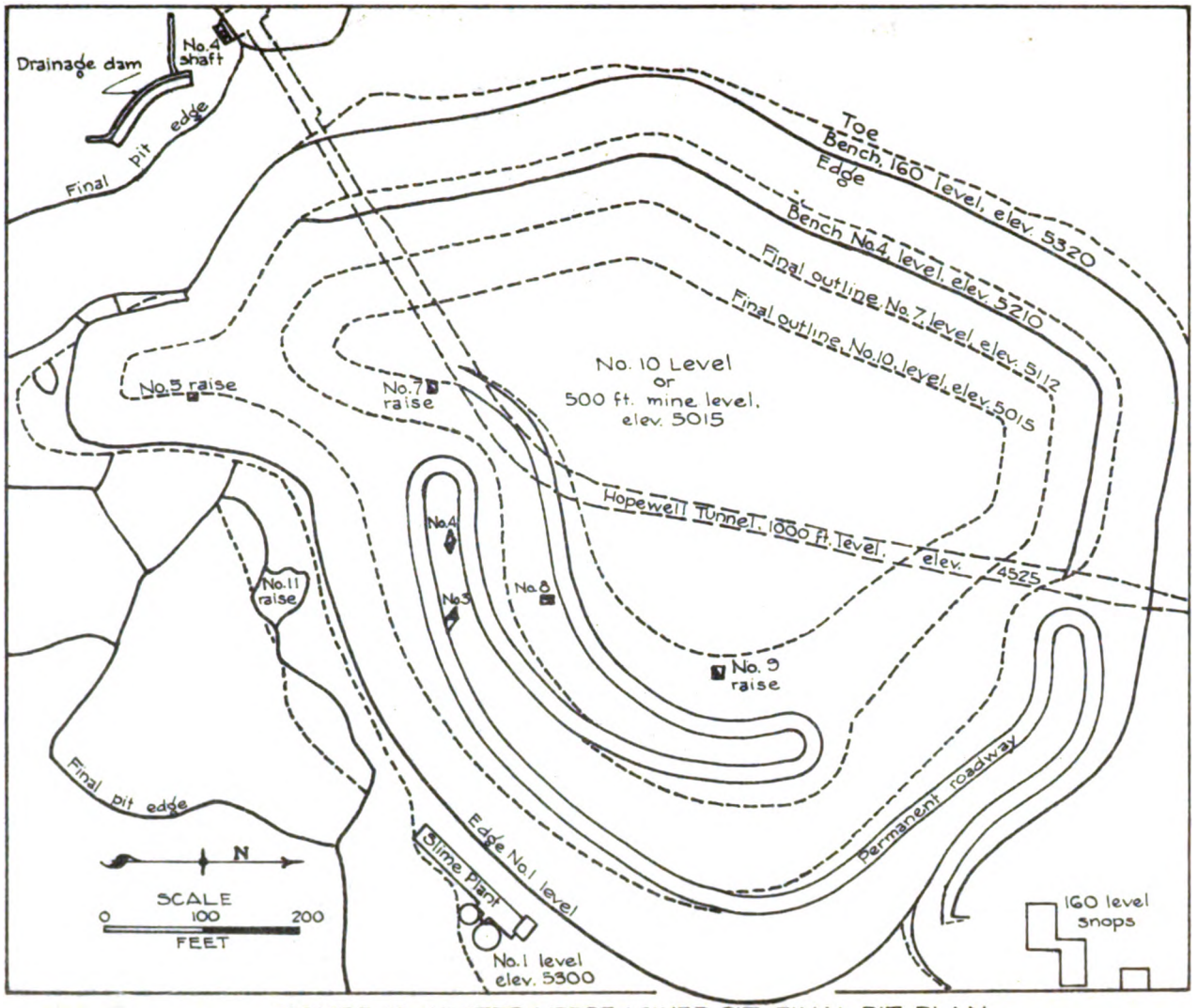

FIGURE 12.-UNITED VERDE LOWER PIT, FINAL PIT PLAN

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removed first, leaving the harder material clean. Old timbers are set aside when encountered and are later loaded into trucks and hauled out of the pit. By planning the work in advance the levels can be developed properly to assure a constant production of the desired material. The small shovels can be quickly moved from one area to another or used between levels if necessary.

Maximum flexibility is possible in this method of operation. Shovels can be worked on any exposed bank. The trucks can be routed from any position to any raise. By this flexibility the pit is enabled to supply any material called for.

By operating on low banks much danger is removed. Men on the shovels and trucks are off of the hot ground and in motion so that they do not have to remain in the gassy areas. Most drilling in fire areas or gassy places is done with churn drills, which likewise can be moved anywhere on the various levels. With the churn drills it is not necessary for the drillers to remain steadily in the bad places.

The lower pit between the 160 and 500 levels is divided into twelve benches varying from 25 to 33 feet in height (fig. 11). At the present writing the first level below the 160 level, or No. 1 level, had been completed. Operations are now being conducted on Nos. 2, 3, 4, and 5 levels, the last of which is 33 feet below the 300 level at an elevation of 5,178 feet.

No. 1 level was first developed by a through cut made by one of the standard 4-cubic-yard steam shovels. The material, which was loaded into dump cars and hauled out, had to be disposed of as encountered; no flexibility was provided. This through cut was started several hundred feet outside of the pit limit on the 160 level on a 3 per cent downgrade and was cut alongside Nos. 3 and 4 raises. The first small shovel was then put into operation. Four light aviation-type army trucks were first used, until the practicability of the plan could be established. These light trucks were unable to pull upgrade. The cut, in going down to No. 2 level, was made in several 7-foot layers, and the shovel loaded the truck on the bench above. This made an upgrade necessary. Before No. 3 level was developed, heavy six-wheel, four-wheel-rear-drive trucks were obtained which could easily pull a load up a 10 per cent grade.

In general, when starting a new cut an attempt is made to confine it within one class of material. The cut is started toward one of the transfer raises so that when it reaches the elevation of the lower bench it will be adjacent to the transfer raise and the shovel can cast directly into the raise. Material can then be disposed of without hauling uphill. As soon as the level is established, work is concentrated in the direction of another transfer raise, and the material is put in the first raise. If a different class of material is encountered, the raise is cleaned out to accommodate that material. Levels are developed in ore first. The first cut is temporary, but a permanent roadway is to be made on the footwall side extending from the 160 level to the successive levels. This road has been completed to No. 4 level (fig. 12). The permanent road is to be on a 10 to 12 per cent grade and will wind back and forth on the footwall side of the pit. As the permanent road can not be conveniently situated with respect to the transfer raises, it is impossible to develop new levels from the roadway.

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Equipment

The equipment used at the United Verde lower pit is tabulated as follows:

Equipment employed at the lower pit

Item and number used	Type	Description
4 Electric shovels	Full revolving	1¾ cu. yd. dipper.
10 Trucks	6-wheel, 4-wheel rear drive	4-cylinder, 10-ton capacity.
1 Truck	do.	6-cylinder, 10-ton capacity.
6 Tractor trucks	2 front wheels, caterpillars on rear	6-cylinder, 10-ton capacity.
2 Trucks	4-wheel	4-cylinder, 5 to 6 ton capacity.
1 Truck	4-wheel	6-cylinder, 3½-ton capacity.
1 Truck	4-wheel	6-cylinder, 2-ton capacity.
1 Truck	4-wheel	6-cylinder, 1-ton capacity.
1 Crane	Full revolving traction	Gasoline-powered.
1 Road scraper		Tractor-powered.

Electric-powered shovels, for which electricity is obtained from the smelter power plant, are used in the lower pit. A considerable saving is made by the use of electricity, as this form of power is more adaptable to the working conditions. These shovels are operated on 3-phase, 440-volt, 60-cycle alternating current. The shovel is equipped with a 100-horsepower hoist motor, 40-horsepower boom motor, and 60-horsepower swing motor. The shovels were originally equipped with 40-horsepower swing motors, but they did not stand up under severe operation. The shovels are mounted on caterpillars having a 36-inch tread and an overall width of 13 feet, 6 inches. The efficiency of the shovels was greatly increased by equipping them with manganese steel caterpillar pads. Other improvements have been made at the United Verde shops such as changing the propelling clutches and installing an automatic air valve for operating the clutch brake ram. The shovel crew consists of three men - an engineer, an oiler, and a pitman. During 1928 an average of 450 cubic yards, or approximately 1,050 tons, per shovel shift was obtained. Power consumption amounted to 0.974 kilowatt hour per cubic yard loaded.

The tramming trucks and tractor trucks are equipped with a standard gravity side-dump body, designed and built by the United Verde Copper Co. By releasing the latches the load can be dumped very quickly when the truck is slightly tilted. The body is built of heavy metal plate, channels, and angle bars riveted together. A double-plate bottom with a wood filler is provided for reducing the shock caused by dropping the heavy material into the truck. The body weighs 3 tons and has a capacity of 5 cubic yards water level, or 3½ cubic yards solid material. The load varies between 7 and 10.5 tons, depending on the character of the material handled. Sulphide ore is estimated at 10.5 tons per truck load; siliceous sulphide 9.5 tons; schist and slag 8 tons; quartz and porphyry 7.5 tons, and diorite, greenstone, oxide, talc, and dike rock at 7 tons. All tramming trucks have been especially

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reenforced to stand up under the heavy loads. They are equipped with steel cabs, hood and radiator protectors. The 6-wheel trucks (fig. 13) have compound transmissions and solid rubber tires. The tractor trucks are also equipped with compound transmissions and in addition have a high-speed reverse which is very useful, as a good deal of backing up is necessary.

Considerable experience has been derived from the trucking problem in the United Verde pit. So far the 6-wheel, 4-wheel-rear-drive trucks have proved to be the most satisfactory. A better distribution of the load is possible, and the flexible arrangement of the rear end protects the frame from undue strain. Good traction is also provided, and the truck is able to negotiate all roads, except on grades above 10 per cent and when the roads are extremely muddy. Rubber wear on the 4-wheel trucks has been found to be much greater than on the 6-wheel trucks. The tractor trucks are useful only for keeping operations going under extremely bad conditions, such as in deep mud and up steep grades. They are especially valuable during bad weather, but can not compare with the 6-wheel trucks under fair conditions.

The roads in the United Verde pit are bad. In fact, the levels are kept as clean as possible so that a truck can run anywhere. The grades are steep, uneven, and frequent. The heavy loads handled and conditions due to occasional bad weather subject the United Verde trucks to extremely severe service. Nevertheless, the situation is being met very successfully.

A truck repair man for making minor repairs is kept in the pit on each shift. For large repair jobs the truck is removed from the pit and put into the shop.

During the year 1928, 167 cubic yards or approximately 390 tons of material were trammed per truck shift. The average length of haul is 500 feet one way. Gasoline consumption amounted to 1 gallon for 11.69 cubic yards trammed. For each shovel shift 2.70 truck shifts were required to keep the shovels operating at full speed.

The miscellaneous light trucks are used for various service around the pit. The 5 to 6 ton trucks are used for handling drill steel, churn-drill bits, casing, and heavy material. The 3½-ton truck is equipped with an acetylene burning outfit and is used for transporting tools and parts for repair work in the pit. The 2-ton truck hauls supplies from the warehouse and the 500-level shops to the pit, as well as transporting the dump crew to Hopewell when it becomes necessary to shift the track on the dumps. The light 1-ton truck delivers powder and blasting supplies where needed in the pit, as well as drilling material.

The gasoline-powered crane is used in doing repair work in the pit as well as for handling heavy material; as it is equipped with a ½-cubic yard clamshell bucket it can be conveniently used for small excavating work in the pit and elsewhere.

The road scraper assists in keeping the roads clean in the pit and around the mine plant.

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One of the 85-ton locomotives, 10 of the 25-cubic yard air-dump cars, and one of the 4-yard steam shovels from the major stripping operations are retained for reclaiming material placed on the dumps during this first work.

Miscellaneous shops, a transformer house, gasoline storage tanks, and a change room have been erected adjacent to the pit on the 160 level.

Drilling and Blasting

General. - In the lower pit care must be taken to break the material clean to avoid dilution. The depth, spacing, and location of holes are dependent entirely on the character of the material to be blasted. Three types of drills are used - jackhammers, a drifter mounted on a tripod, and electric traction churn drills. In breaking ground along contacts, in inaccessible places, and in bank-trimming the air drills are used; however, in uniform material and in hot ground the churn drills are preferred. In the lower pit, actual burning sulphides in the underground fire zone were encountered, which presented a difficult blasting problem.

A comparison made on the cost of breaking ground by the use of air drills and churn drills showed that although the cost of powder consumed in blasting churn-drill holes was 50 per cent more, a considerable saving was made in labor, power, and repairs. The total cost per ton broken with the churn drill was found to be 16 per cent less than with the air drill.

Air Drills. - Jackhammers are used to drill toe holes and vertical holes. Methods and equipment are similar to those employed in the major stripping operations, except that the jackhammer is an improved type.

The heavy drifter on a tripod mounting is used only for drilling toe holes in the very hard sulphide which can not be drilled with the jackhammers.

Air-drill holes are loaded with 35 or 50 per cent strength gelatin dynamite. The charges are fired with No. 8 electric detonators from the 440-volt line. The use of the powder-loading machine was abandoned, partly on account of irregularity in spacing and chambering of holes, but largely because of the hot, steaming ground encountered, which made the use of the granular powder unsatisfactory.

No "dobie" blasting is done in the pit. All bowlders are block-holed. The cost of bock-holing is about 2 cents per ton of material broken.

Churn Drills. - Four electric-powered churn drills are used in the United Verde pit in place of the old steam drills. Fuel cost on the steam drills amounted to $8 a shift against a power cost of $0.60 a shift on the new drills. Other improvements such as a wire-rope drilling cable favored the use of the new drills. A churn-drill crew consists of a driller and helper.

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Churn-drill holes are drilled on the edge of the bank. Drilling speeds vary from three-fourths of a foot to 8 feet per hour. Holes are drilled 5 feet more in depth than the height of the bank. If the burden is too great, machine-drill toe holes are used with the churn-drill holes. Holes are spaced according to the conditions but are generally 10 to 12 feet apart. However, in hot ground, where it is impossible to place the desired amounts of explosive in the hole, the holes are spaced 4 to 5 feet apart. Hot and gassy ground is generally drilled with churn drills, as the larger hole can be more quickly and safely loaded with a greater amount of gelatin. Also fewer men have to be kept on the bad ground.

As a rule between 150 and 250 pounds of 35 per cent or 50 per cent straight gelatin dynamite are used in a hole. Some granular ammonia dynamite is used where it can be kept dry. Holes are generally tamped with sand. Blasting switches are located conveniently on the upper levels, and the blasting wires are wound on a reel adjacent to the switch. From these reels the blasting wires can be run out to any location in the pit.

Liquid oxygen explosive was experimented with in blasting churn-drill holes. It was found to effect a considerable saving, but because of insufficient knowledge of the action of "LOX" in possible hot ground, it was decided to be unsafe to continue its use.

Blasting Hot Ground. - In holes where heat is encountered a potentiometer with an iron-constantin thermocouple is used to measure the temperature. The highest temperature that has been measured in a hole is 780°F. A temperature of 120°F. is considered safe for loading and tamping a hole, using the ordinary methods. If cooling a hole is possible, the cooling is accomplished by running water into the hole for a period of 1 to 24 hours, or by using sand. By dropping sand in the bottom of the hole, the heat has been reduced in several instances from 212 to 120°F. long enough to permit loading and firing. When sand is used the hole must be chambered sufficiently to hold both sand and explosives. Sand is placed in the hole with a small amount of water, which washes the sand into the crevices around the chamber. The hot gasses are sealed off and the sand does not become heated readily as it is a poor conductor of heat. As many as three wheelbarrows of sand have been placed in some holes.

For holes that can not be cooled, a torpedo is made which consists of an insulating cartridge filled with explosives (fig. 14). The torpedo used in small machine-drill holes consists of an ordinary mailing tube filled with gelatin dynamite. For the larger 6-inch churn-drill holes a large paper tube with a 4-inch inner diameter and $\frac{1}{2}$-inch wall, 6 to 8 feet long, is used. A wooden plug is driven in one end and a wire by which the tube can be lowered into the hole is fastened to it inside the tube. The charge and detonators are placed in the tube and covered with sand. In shooting a round of hot holes the torpedoes are wired and connected with the blasting line and placed off of the ground adjacent to the holes. When the blasting signal is given, the torpedoes are immediately lowered into the holes so that they can be detonated as soon as the men are in the clear. During the short interval that the torpedoes are in the hot ground the heat is not able to penetrate the tube. These hot holes are not chambered nor is tamping used.

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Thirty-five per cent gelatin dynamite and some granular ammonia dynamite are used in the torpedoes. Care must be taken that the granulated powder does not leak out of the torpedo.

Hot ground can not be blasted efficiently, as enough powder can not be put in the torpedo and what is in the torpedo is not in a chamber nor tamped. The use of the small torpedoes for the machine-drill holes is particularly inefficient. However, it is necessary to blast the hot ground in this manner, and a fair degree of success is being obtained. Accidents caused by blasting hot ground have been eliminated by the use of the torpedo.

Pit Operation

At present (November, 1929) 21,000 tons of direct-smelting ore and 3,500 tons of converter ore are mined weekly in the pit, and milling ore is mined as called for. In addition, the low-grade material which is placed on dumps at the mouth of Hopewell Tunnel is handled. It is estimated that 1 ton of low-grade material should be removed from above the No. 5 pit level for each ton of direct-smelting ore mined in order to make the ore available below this level.

Every morning the pit foreman is informed from the smelter and by the foreman of the tunnel transportation what the pit requirements for the 24 hours will be. This information is passed on to the other shifts so that the order can be completed. At the beginning of each shift the crews are told to what shovel, level, and place they are consigned, and are moved about during the shift if any changes arise. Truck drivers and dumpers are assigned to certain shovel crews and raises. Enough trucks should be available to keep the shovel busy at all times. On a short haul two trucks are sufficient, but as many as six trucks are necessary for some of the longer hauls. The truck dumper is stationed at the raise. It is his duty to keep the dump clean, adjust the safety chains on the trucks, see that the dump body is properly lubricated, and keep a record of the loads dumped.

During the shift the pit foreman is kept informed by the tunnel foreman as to the condition of the raises. Close cooperation has to be maintained in order to prevent the material from hanging up in the raises. The material should not be allowed to pile up in the raises above the grizzlies, as the larger pieces are likely to form an arch and the fine material often cements. Also if a heavy rain occurs and the material in the raise becomes muddy, a tremendous hydrostatic pressure may be developed. In one instance the pressure on being released forced out the chutes of the ore bin and flooded the haulage tunnel.

Clogging of raises with large rocks has been almost eliminated by not allowing any piece to be loaded which will not pass through the shovel dipper. However, the larger pieces are often held up on the grizzlies and the material starts to back up in the raise before they can be broken. An attempt is made to mix the coarser material with the finer in order to prevent choking the raises by the cementing and packing of the finer material.

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The transfer raises are accessible on the mine levels, but are sealed off by doors and bulkheads from the mine workings. In case a raise becomes clogged, all dumping into that raise is stopped as soon as the condition is discovered. The raise is then inspected on various levels to determine where the choking occurred. Generally the material can be loosened by blasting in the raise. At times it has been necessary to tie as many as three or four 20-foot tamping sticks together with a charge placed on the end, and push it up the raise where the material is hung-up. On several occasions material has been loosened by allowing water to run down the raise which washes out the finer material wedged between the larger rocks.

At present eight shovel crews are maintained, three each on the day and afternoon shifts and two on the night shift. A spare shovel is available during the day for the repair crew to work on.

During 1928 the locomotive crane equipped with a clamshell bucket was used on the 160-level bench below the high diorite bank for cleaning up the material that slid off the bank. The loaded cars were dumped into the main mine waste raise. During the year 14,886 cubic yards was removed by the crane on this bench.

The day shift begins at 8 a.m., the afternoon shift at 4 p.m., and the night shift at midnight. Blasting is confined to regular hours - noon, 4 p.m., 8 p.m., and midnight - so that there will be a minimum of delay.

Sampling

A sampler who works on day shift samples the banks each day after they are cleaned up. The assay results of the samples are given to the pit foreman and powder foreman who plan the work accordingly. Drill holes are occasionally sampled but generally the class of material can be accurately determined from a knowledge of the geology and the value of the face of the bank. Samples are also taken from the broken material ahead of the shovel by the assistant samplers; one sample is taken for each 10 trucks loaded. A record is kept of the material removed. The bank sample assays are mapped for use in estimating the character of the material on the lower benches.

Disposal of Material

After the material is dropped into the raises it is hauled out of the tunnel and dumped into transfer bins at the entrance of Hopewell Tunnel. There the direct-smelting, converter, and concentrating ores are transferred to large cars to be hauled to the smelter. Low-grade material is placed on various dumps at Hopewell; separate dumps are maintained for leaching, low-grade converter, low-grade sulphide, and waste. Most of the waste from the pit is dumped into a waste transfer raise from which it can be distributed through the underground mine for stope filling. However, with the increase in production it was necessary to place some waste on the dump in order to make the ore available.

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Classification of Pit Material

Material in the United Verde pit is classified according to the following schedule:

I. Converter Ore.

A. Smelter shipments.

1. Aluminous converter material, such as quartz porphyry to contain a minimum of 60 per cent silica. Alumina present must not exceed one part to 5.5 parts silica. Values to exceed $4 a ton.

2. Oxide and quartz converter material to contain a minimum of 50 per cent silica. Values to exceed $4 a ton.

3. A small amount of sulphide material will be permitted if silica content exceeds 65 per cent silica and values $4 a ton.

B. Hopewell stock-pile shipments (Dump No. 34).

1. All converter material with value less than $4 a ton.

II. Direct-Smelting Ore.

A. All sulphide material containing a minimum of 1.0 per cent copper. Basic and siliceous sulphide material to be mixed so that the average silica content of the ore received at the smelter is about 23 per cent.

B. Low silica content oxide and old filling containing a minimum of 2.0 per cent copper, or 1.5 per cent copper when gold and silver value is in excess of $1 a ton, or no copper value when gold and silver value is more than $4 a ton may be added to the sulphide ores, provided that the mixture does not exceed 23.0 per cent silica, nor more than 1 part alumina to 5.5 parts silica, when received at the smelter.

III. Concentrating Ore.

A. Clean schist or porphyry having 1.0 per cent copper or better. Low-grade ore should be mixed with high-grade ore so that the resultant mixture will approximate 3.0 per cent copper. Under no circumstances will filling be mixed with the concentrating ore.

B. A small amount of clean siliceous sulphide with copper values between 1.0 and 5.0 per cent may be mixed with the schist and porphyry.

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IV. Leaching Material (Dump No. 36).

A. All low-grade copper-bearing material between 2.0 and 0.7 per cent copper, excepting material suitable for smelting, milling, and the converters.

V. Low-Grade Sulphide (Dump No. 37).

A. All sulphide material below 1.0 per cent copper not included in the above.

VI. Waste.

A. All barren material.

As no bedding bins exist in the Clarkdale smelter it is necessary for the pit department to control the silica and alumina content of the direct-smelting ore. The direct-smelting ore mined in the underground mine also has to be taken into consideration when determining the proper mixture. It is therefore necessary for the pit to determine the proportion of basic, siliceous sulphide, and stope filling which enters into the smelter mixture. The mixture that arrives at the smelter should contain about 23 to 24 per cent silica. If the mixture is off one day, an attempt is made as soon as possible to correct it by adding more basic or siliceous sulphide ore, as the need may be.

The United Verde pit differs from most pit operations in that it is producing a high-grade ore. The direct-smelting ore mined averages about 5 per cent copper, while the concentrating ore averages approximately 3 per cent and the converter ore 2.5 per cent copper.

Probably half of the stope filling encountered is shipped to the smelter as ore. The copper in the stope filling is due to the following causes:

1. In the early days of the mine, low-grade material which was not commercial at the time was thrown in as filling, especially low-grade schist which is hard to smelt.

2. Early mining methods were not clean. In several of the exposed stopes enough high-grade ore is scattered through the filling to make it all run as ore at the present day.

3. Enrichment has resulted from deposition of copper sulphate and copper carbonate by water descending from the old workings. Enrichment has also been caused by gases ascending from the fire area and depositing copper salts in the filling.

4. Some of the sulphide in the fire areas has also been enriched by the removal of sulphur by roasting.

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Dilution

An attempt has been made to estimate the percentage of dilution of the pit ore. The problem of estimating dilution is difficult, however, because unknown areas of ore are uncovered and because a large portion of the old stope filling is being shipped as ore.

A summary of the United Verde lower-pit operations for 1928 follows:

Table 3. - Summary of costs

United Verde Lower Pit Operations — Year 1928

Cubic yards stripping loaded	322,538
Cubic yards ore loaded	276,055
Total cubic yards loaded	598,593
Dry tons ore loaded	731,488

Direct operating costs per cubic yard based upon total yardage (ore and stripping)

	Labor	Comp. air[1] drills and steel	Power cost	Explosives	Repairs[2]	Supplies[3]	Miscellaneous	Total
Shovel operation	$.047	- -	$.012	- -	$.117	$.001	$.003	$.180
Drilling and blasting	.128	$.084	.002	$.082	.- -	.027	.004	.327
Truck tramming .	.055	- -	- -	- -	.113	.019	- -	.187
Total operation.	$.230	$.084	$.014	$.082	$.230	$.047	$.007	$.694

1 Includes repairs to drills and sharpening steel and drill bits.
2 Includes labor, supplies, and power, used in repairing.
3 Includes gasoline and oil used on trucks.

The direct costs of shovel operation, drilling and blasting, and truck tramming are divided against stripping and mining in proportion to the tonnages of the two classes. The direct costs for stripping and mining for 1928 are tabulated below:

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Direct costs for stripping and mining, 1928

	Per cubic yard	Per ton
Stripping (322,538 cu. yds.):		
Shovel operation	$.161	- -
Drilling and blasting	.293	- -
Truck tramming	.160	- -
Dump expense	.038	- -
Miscellaneous	.018	- -
Total	$.670	- -
Mining (276,055 cu. yds. or 731,488 tons)		
Shovel operation	$.204	$.077
Drilling and blasting	.368	.139
Truck tramming	.219	.086
Total	.791	.302

General data for 1928 are tabulated as follows:

Total 8-hour shovel shifts	1,285.75
Total 8-hour truck shifts	3,471.13
Total 8-hour churn-drill shifts	1,384.83
Total churn-drill footage	37,812
Feet per shift	27.3
Pounds of explosives per cubic yard	0.483
Total man-shifts	48,374
Cubic yards per man-shift (total material)	13.21
Tons per man-shift (total material)	32.62
Labor cost per cubic yard	$.3678
Operating supplies per cubic yard	$.1137
Repair supplies per cubic yard	$.1291
Total supplies per cubic yard	$.2428
Total cost of repairs per cubic yard	$.2728

RECLAIMING DUMPS

As converter ore is not abundant in the lower pit it is necessary to ship some of the converter material from the dumps which were established at the time of the major stripping operations. The material is loaded by a standard 4-yard shovel directly into cars which transport the material to Clarkdale. This is not a continuous operation, but the shovel is steamed up and the material loaded when called for. The cost for loading 39,772 cubic yards or 67,175 tons from the dumps for the year 1928 is tabulated as follows:

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	Per cubic yard	Per ton
Shovel operation	$0.093	- -
Locomotive operation	0.074	- -
Drilling and blasting	0.041	- -
Miscellaneous	0.084	- -
Total	$0.292	$0.173

LEACHING

Water from the mine is run through the dump of leaching ore and takes some of the copper in the low-grade material into solution. The solution is gathered in ditches at the bottom of the dump and run over scrap iron in precipitating troughs, where the copper is recovered. The copper precipitate recovered contains about 80 per cent copper.

LABOR

In November, 1929, 225 men were employed in the United Verde pit. At present they are paid a 15 per cent bonus above a base scale. This bonus is based on the price of copper and on general operating conditions.

An efficiency bonus based on the work performed is also paid for most classes of labor. Standards are set for each class of work and a record is kept of the performance of each man or group of men. The men are paid on a basis of one-half of the work performed in excess of the standard amount. For instance, if the efficiency is 150 per cent, the workman receives 25 per cent of his base rate as a bonus. In some cases the bonus is figured on the basis individual work; in others the performance of a group is considered.

ADMINISTRATIVE ORGANIZATION

The general form of organization of the United Verde Copper Co., with more detailed form of the pit organization, is given in Table 4.

Cooperation between the mine departments, as well as the smelter departments, is maintained by the pit superintendent. An engineer is employed to supervise the engineering, sampling, and clerical work in the pit. Assistance in engineering and geological work is furnished by the main office. Pit records are kept in the pit office. The pit office also assists the main time office and safety department.

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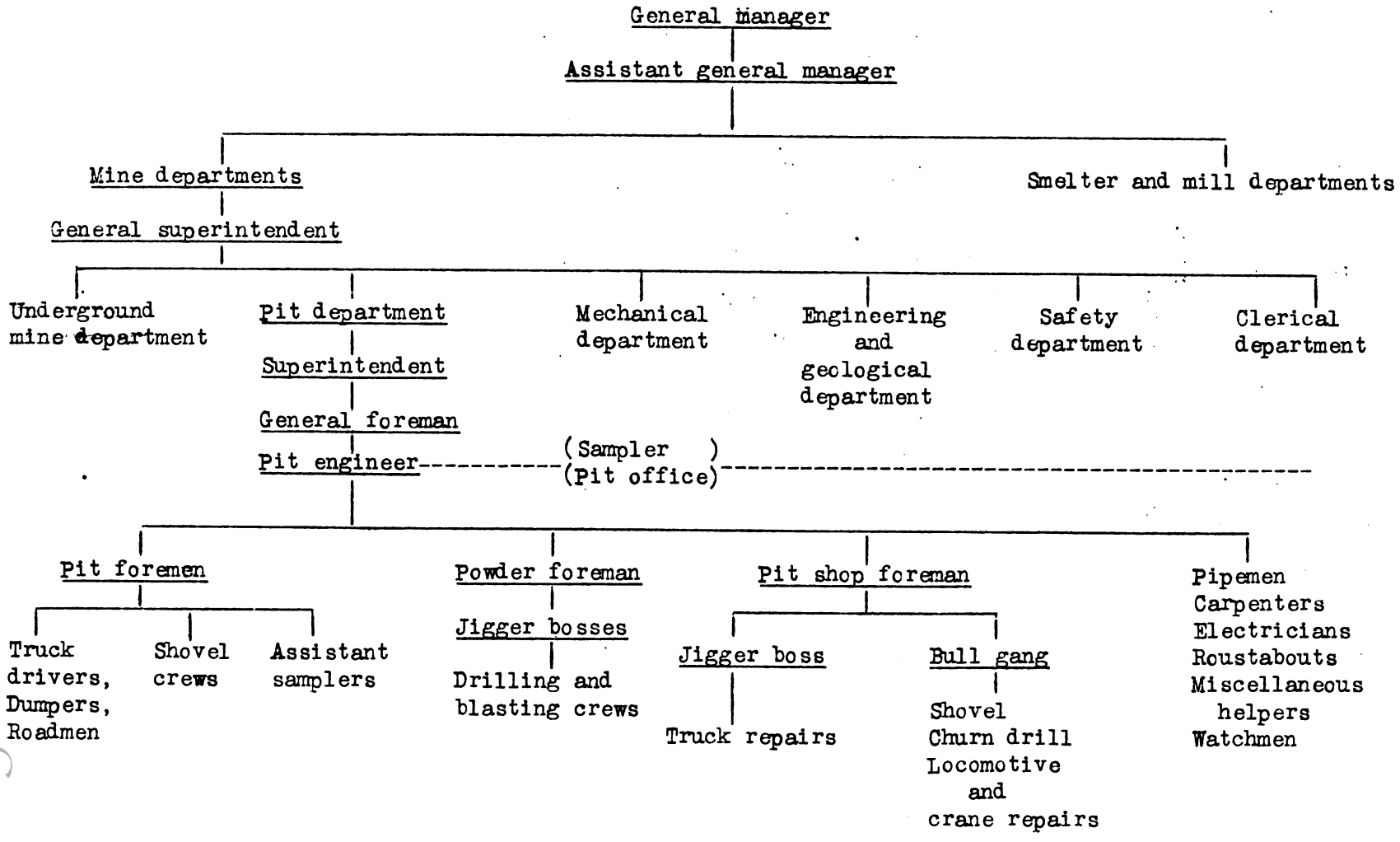
TABLE 4.- Administrative organization of the United Verde open-pit mine



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SAFETY

Safety work in the pit is supervised by the safety engineer in charge of the entire Jerome operations. Hard-toe shoes are required, and goggles must be worn when performing certain classes of work. Close co-operation has to be maintained between the working sections of the mine, especially in the matters of blasting around raises and allowing men to be in underground workings which might be disturbed by the pit operations.

During 1938 five lost-time accidents occurred, resulting in a loss of 101 shifts, or of 1 shift for each 479 shifts worked.

MINE FIRE

An enormous amount of sulphur gases and smoke from the underground fire was encountered as the ground was dug in advancing the first bench below the 100-foot level on the west side of the pit. Respirators, gas masks, and extra crews were tried without success. Part of the pit had to be abandoned for a while.

In some areas the actual sulphide ore was burning; in others only the timbers started burning when exposed to the air. The timbers in some of the old stopes were completely burnt out, and the filling had cemented and formed casts where the timbers had been.

In the early part of 1937 it was decided to use slimes to seal off the fires. Tailings from the mill at Clarkdale were used with immediate success; no part of the pit is now uncomfortable to work in. The tailings were poured on the surface and pumped into churn-drill holes in the fire areas, thus filling the cracks and crevices from which the gas and smoke escaped and sealing off the air. In order to eliminate the high cost of transporting the tailings up the steep grade from Clarkdale, a sliming mill was constructed adjacent to the pit. A drift was run into the quartz porphyry and a glory hole started to obtain material. The slime plant consisted of a jaw crusher, rolls, ball mill, and storage tanks. The capacity of the mill was approximately 65 tons solid per shift. The slime consisted of 50 per cent moisture. The solids were ground so that about 75 per cent was minus 100 mesh.

The slimes were later pumped into the underground fire areas through the bulkheads which sealed off the fire country and assisted materially in reducing the mine fire.

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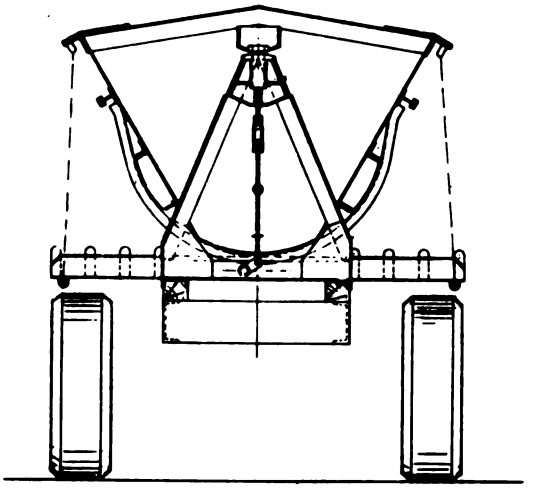

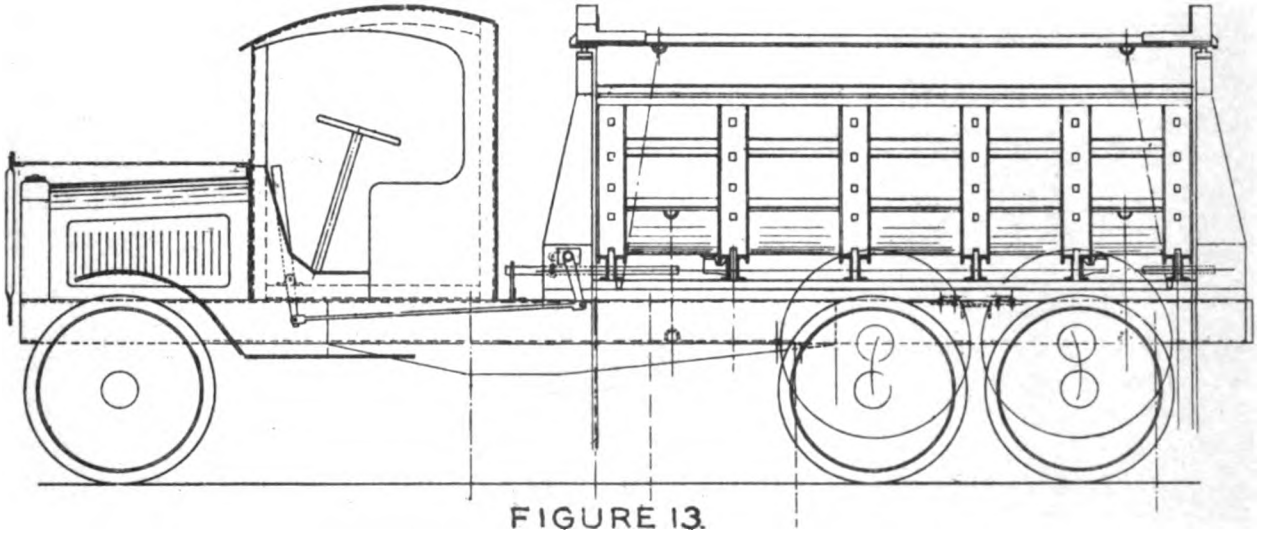

FIGURE 13.
5-CUBIC YARD 6-WHEEL TRUCK

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